MINING engineering

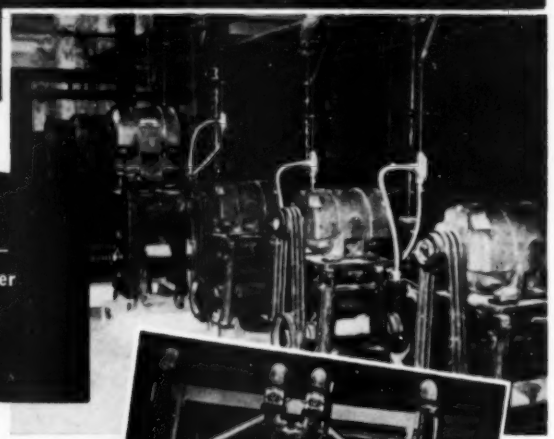
MARCH 1953

DIATOMACEOUS EARTH • TIMBER HANDLING • SLIDING SCALE IMPORT TAX

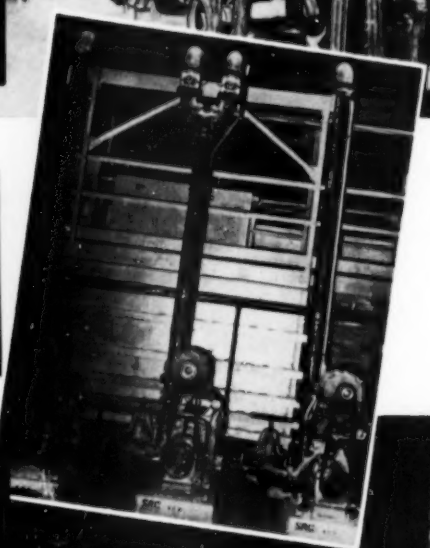
WILFLEY *Sand* PUMPS

In the World's Greatest Open Pit Copper Mine and Sulphide Plant at Chuquicamata, Chile

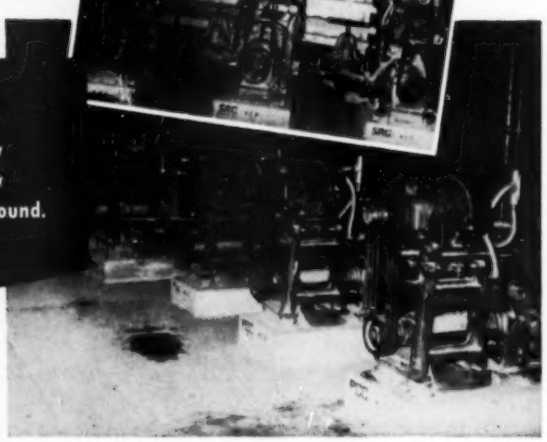
Wilfley Sand Pumps **are an integral part of the huge operation carried on in the fabulous new $100,000,000 plant at Chuquicamata, Chile. Day and night dependable Wilfley's deliver continuous, trouble-free performance . . . playing a major role in the dramatic success of this great mining enterprise.**



Primary Rougher Overflow Pumps—Secondary Rougher Overflow Pumps in background.



Secondary Cleaner Tails Pumps—Secondary Cleaner Overflow Pumps in background.

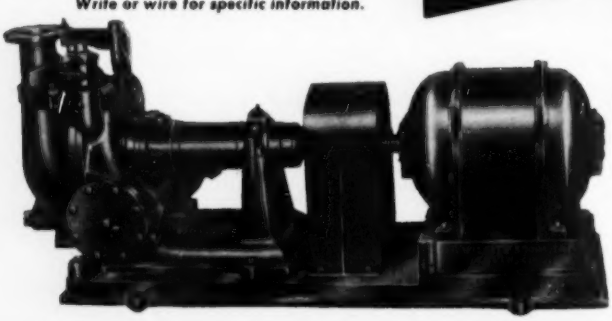


Final Concentrate Pump—Secondary Rougher Overflow Pumps in background.

New Model "K" WILFLEY *Sand* PUMP

The new Model "K" Wilfley Sand Pump answers the 3 important economic considerations in the handling of solids: **1.** High Operating Efficiency—the Model "K" features important mechanical improvements which give you substantial dollar savings in power and production costs. **2.** Low Maintenance Cost—Wilfley's precision engineering assures dependable operation with a minimum replacement of parts. **3.** Minimum Installation Expense—the Model "K" is specially designed for simple installation, saving plant operator's time, trouble and added expense.

Write or wire for specific information.



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Denver, Colorado, U.S.A.

New York Office: 1775 Broadway, New York City, U.S.A.

MINING engineering

VOL. 5 NO. 3 MARCH, 1953

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COVER

The cover is a micrograph of diatomaceous earth from Palos Verdes, Calif. This particular stratum contained only needle-like diatoms. In billions this microscopic aquatic plant with a skeleton becomes big business, as described in the article on page 277.

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— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Metallurgical Engineer, 32, married, B.S. mineral dressing. Experience in open-pit mining, ore dressing, sintering as superintendent and assistant plant manager of large operation. Good background in cost reduction, report preparation, maintenance and plant operation. Desires responsible administrative position. M-12.

Mining Engineer, 31, married, children. Four years' mining experience in production, development, and exploration. Three years' experience in timber and steel construction. Desire supervisory position in development, exploration work or production mine foreman. Will accept foreign position single status for limited time only. Present location Washington. Available reasonable notice. M-13.

Mining Engineer, 30, married. Experienced underground engineering and surveying, supervision of mine plant construction, development planning, and housing project development. B.S. mining, and ICS course in civil engineering. Available two weeks notice. Prefer western U. S. M-14-518-E-10-San Francisco.

Graduate Mining Engineer, 40, now employed as mine superintendent, desires management or executive position United States. Sixteen years' experience hard rock mining, shaft sinking, and complete modernizing of reopened property producing 2,000 tons daily. Proven production ability in mines and an industrial plant with difficult labor and supply conditions. M-15.

POSITIONS OPEN

Assayer, experienced in fire and wet assays, for assays consisting of silver, gold, lead, zinc, iron and solubles, manganese and copper. Single man required. Salary, $3000 a year plus room, board and transportation. Location, Central America. Y8189.

General Manager with technical education and experience and executive ability to assume complete responsibility in operating a plant doing stripping, mining, hauling, milling, drilling and delivery of manganese ore. Location, Arkansas. Y8066.

Mining Engineer with considerable exploration and development experience for mine examination work with consulting firm on foreign and domestic projects. Salary open. Location, New York, N. Y. Y8155.

Mining Engineer with an engineering degree and minimum of five years' experience, preferably with open cut methods. Must speak Spanish. Salary open. Location, Chile. Y8138.

Geologist experienced underground water resources. One for twelve months in Turkey; the other for fifteen months in Portuguese Angola. Salaries, $12,000 a year plus all expenses. Y8122.

Engineer. (a) Mine Foreman with at least three years' underground experience for lead-zinc operation. Salary, $4800 a year plus housing and family accommodations. (b) Shift Boss, preferably engineering graduate, with underground experience. Salary, $4200 a year plus housing and family accommodations. Must speak Spanish. Location, Peru. Y6679.

Mine Superintendent for small underground gold and copper mine in western Argentina. Good climate; elevation 10,000 ft. Must speak Spanish. Must accept single status in the beginning. Y8118.

Director of Engineering for an engineering institute, 30 to 40, with Doctorate in mining, chemical, electrical or mechanical engineering. Must have good experience in research, preferably industrial, and a good teaching record. Salary, $6000 to $7000 a year. Location, South. Y7166.

Mining Engineer, B.S., 33, married, one child. Seven years' experience; completed one year training program open-pit and underground; sales service engineer for exports to South America; wide blasting experience; prospecting and exploring in Caribbean and South America. Spanish and Portuguese languages. M-16.

Resident Manager, Superintendent and Mining Engineer for an asbestos project now operating on a small scale in Venezuela. Should be experienced in the asbestos business, for company expanding the mining and milling facilities and the camp generally. Y8180.

DESIGN ENGINEER for plant layout and structural design work on mining and ore handling facilities. Permanent position in Pennsylvania with mining department of large steel company. Salary open. Applicants should be college graduates or equivalent with experience.

Box C-7 MINING ENGINEERING

MECHANICAL ENGINEER, college graduate, for design and office engineering position with a large mining organization. Location Eastern Pennsylvania. Salary open. Applicant must have practical experience with equipment used in the mining industry and should be a competent draftsman.

Box C-8 MINING ENGINEERING

MINING or MECHANICAL ENGINEER to assist in export sales of mining and earth moving equipment. Knowledge of foreign languages helpful. Location New York City. State salary desired.

Box B-2 MINING ENGINEERING

MECHANICAL ENGINEER. Opportunity for experienced mechanical engineer to join long established company in Midwest. Products sold throughout the world. Excellent employee benefits. Must be capable of supervising design and developments of major equipment used in mining, crushing and cement industry operations. Replies treated confidentially. Give experience, age, references and salary requirements.

Box C-5 MINING ENGINEERING

ASSISTANT or ASSOCIATE PROFESSOR of Metallurgical Engineering; Mining School in the East; some mining background desired but not necessary. Required to assist in teaching Mineral Dressing and mainly Extractive and Process Metallurgical Engineering subjects; services required beginning September, 1953; salary depends upon qualifications and experience.

Box C-6 MINING ENGINEERING

Experienced graduate metallurgists and metallurgical engineers required for large research and development organization in Northern Rhodesia. Duties relate to process metallurgy of copper, cobalt, lead, zinc and other base metals, with emphasis on ore concentration by flotation. Applicant must have minimum 5 years' industrial experience. Total remuneration, £2,000 to £2,500 according to qualifications and experience, comprises basic salary, variable metal bonus and cost of living allowance. Generous pension and leave privileges. Housing provided at nominal rental. Free transportation to Northern Rhodesia for successful applicant and immediate family. Applications stating age, marital status, qualifications, experience record, availability, and accompanied by a recent photograph should be addressed to The Secretary, P. O. Box 172, Kitwe, Northern Rhodesia.

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SHOT FIRING CABLE

EVER PRODUCED!

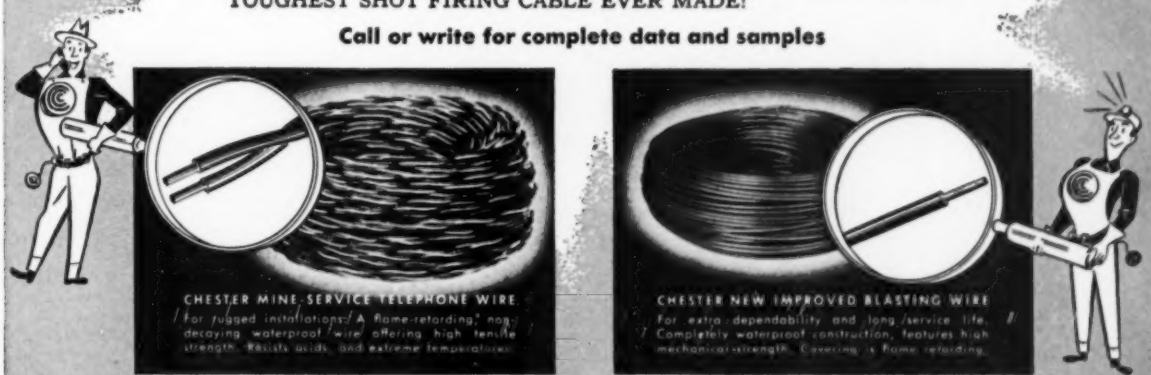
Check the long life qualities offered by this *SUPER DURABLE* **SHOT FIRING CABLE**

- RESISTS SHARP EDGES AND ABRASION
- FLAME RETARDING AND WATERPROOF
- NON-DECAYING AND NON-AGING
- HIGH STRENGTH AND FLEXIBILITY
- NON-CRACKING FROM −40°C to +80°C
- RESISTS MOST ALKALIS, ACIDS, OILS, GREASES
- IMMUNE TO OZONE AND SUNLIGHT

Exclusive CHESTER nylon jacket over center core increases service life and promotes mine safety

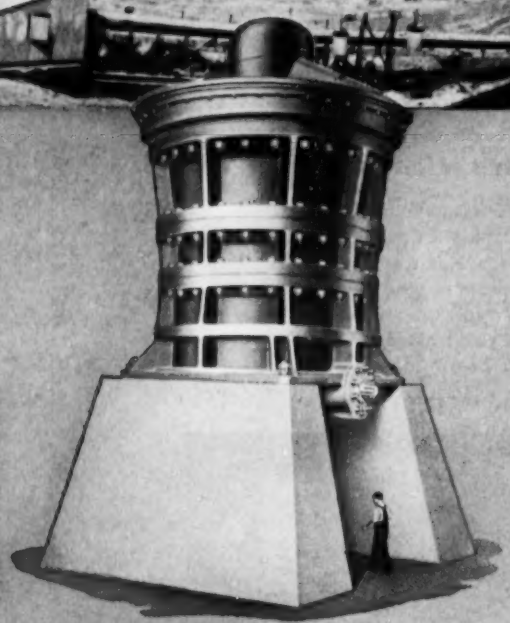
Here's the shot firing cable you can depend on for really tough mine service. It's highly durable when exposed to sunlight and ozone, withstands dragging over sharp edges and abrasives and is highly flexible. Non-cracking over a wide temperature range, Chester is flame retarding and waterproof—the kind of cable that's good for years of usefulness in all types of mining operations. Ask for Chester the next time you order — you'll agree, IT'S THE TOUGHEST SHOT FIRING CABLE EVER MADE!

Call or write for complete data and samples


CHESTER MINE-SERVICE TELEPHONE WIRE
For rugged installations. A flame-retarding, non-decaying waterproof wire offering high tensile strength. Resists acids, and extreme temperatures.
CHESTER NEW IMPROVED BLASTING WIRE
For extra dependability and long service life. Completely waterproof construction, features high mechanical strength. Covering is flame retarding.

CHESTER CABLE CORP.
CHESTER, NEW YORK

MANUFACTURERS OF QUALITY WIRES AND CABLES FOR EVERY INDUSTRIAL AND COMMERCIAL APPLICATION



WORLD'S LARGEST CRUSHER is this 3500-tph, 60-109 *Superior* gyratory being built for Mesabi's first commercial taconite plant. Ultimate plant output will be 10 million tons.

FINES ARE SCALPED from r.o.m. ore on sturdy 1000-tph vibrating screens like this one in Minnesota. A-C builds six separate types of vibrating screens for beneficiation.

EQUIPMENT FOR IRON ORE BENEFICIATION: Gyratory and jaw crushers . . . Vibrating screens . . . All types of washers . . . Concentrating jigs . . . Rod, ball, and *Ballpeb* mills . . . Centrifugal pumps . . . Rotary kilns and coolers . . . *Texrope* V-belt drives . . . All types and sizes of electric motors and generators . . . Starters . . . Transformers and voltage regulators . . . Complete power generation, distribution, control equipment.

the Bottom!

As Mesabi Ores Lean Out, Allis-Chalmers Aids in Beneficiation

DIRECT SHIPPING ORES are fading fast up on the once fabulous Mesabi range. But despite foreign ore discoveries, by 1955 a quarter of blast furnace feed must be beneficiated domestic ore.

Allis-Chalmers — manufacturer of equipment for the steel industry from the mine to the mill — is helping lead the way to more economical beneficiation of lean ores.

Does low ore content require you to triple material handled? Allis-Chalmers will build you bigger, more efficient equipment! At far left below is the world's largest crusher, being built by A-C for a Mesabi taconite plant.

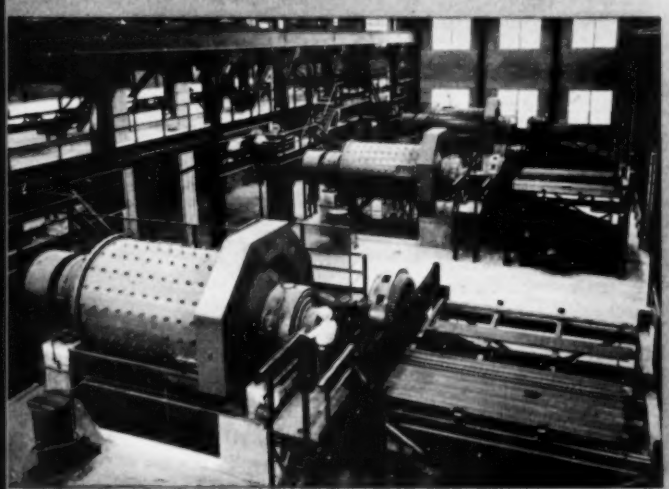
Has proper yet economical pelletizing got you stumped? Allis-Chalmers has just put into operation a pilot Pelletizing and Heat Hardening plant built exclusively for researching new pelletizing methods.

Do you have a beneficiation "shortcut" in mind? Don't let lack of equipment postpone your testing of new processes. The Allis-Chalmers Processing Research Laboratory stands ready to accurately and exhaustively test any processing method on a confidential basis.

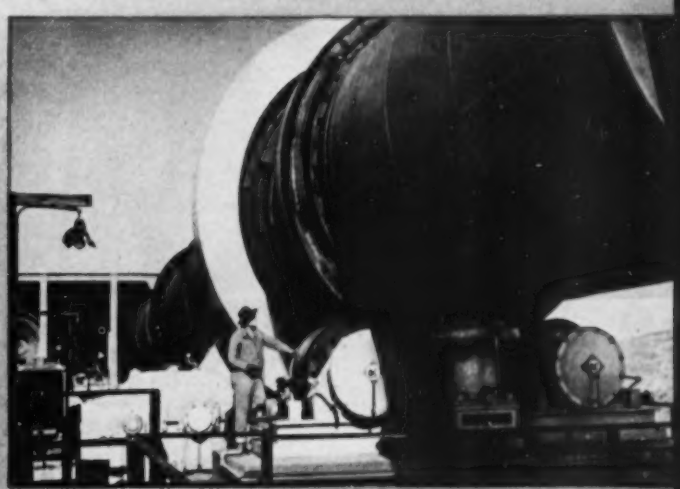
For over half a century Allis-Chalmers equipment has been processing ores for America's steel industry. Some of the cost-cutting, tonnage-increasing ore beneficiation equipment in the broad A-C line is seen on these pages.

Allis-Chalmers offers you *one-manufacturer responsibility* for process, drive and control equipment all along your flow sheet. For literature or detailed information on products and services, call your nearest A-C office or write to Allis-Chalmers, Milwaukee 1, Wisconsin. A-3808

Superior, Ballpeb and Texrope are Allis-Chalmers trademarks.



ORE IS GROUND in three rod and ball mills ahead of magnetic separation at this beneficiation plant. A-C builds *all* types of grinding mills for ore beneficiation processes.



IRON ORE IS AGGLOMERATED to make good blast furnace and open hearth feed in Allis-Chalmers rotary kilns in connection with iron mines and beneficiation plants.

ALLIS-CHALMERS

Power, Electrical, Processing Equipment for Iron and Steel

DRAWPOINT LOADING

MECHANICAL LOADING
SAVES TIME—MONEY
SPEEDS PRODUCTION
INCREASES SAFETY

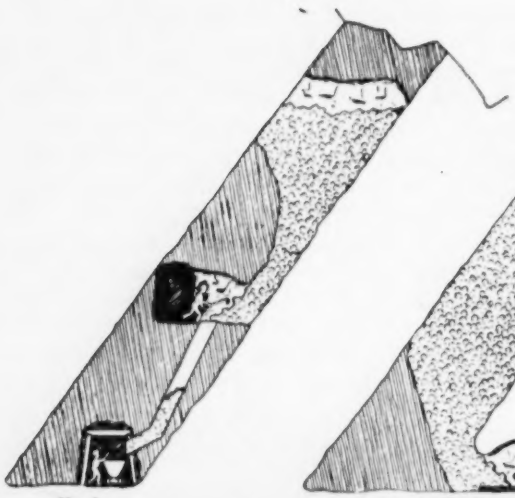


Fig. 1

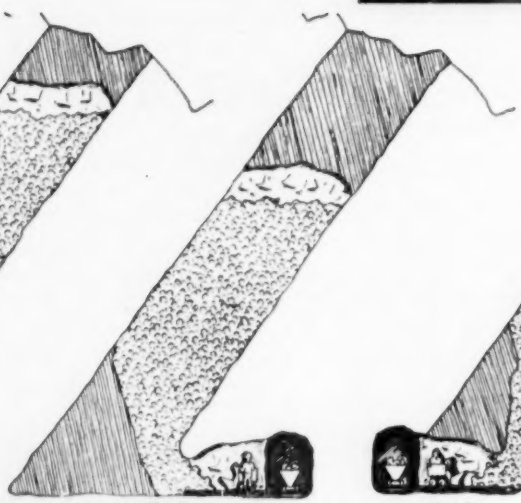


Fig. 2

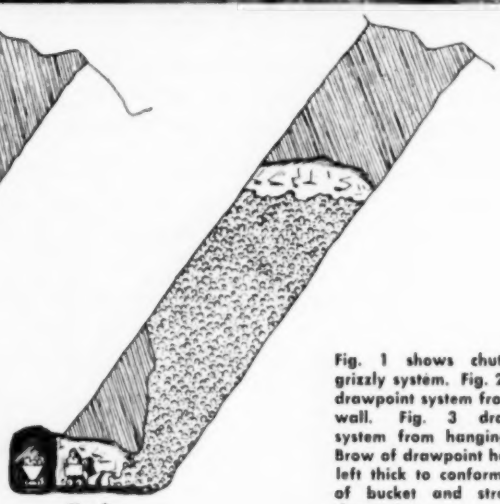


Fig. 3

Fig. 1 shows chute and grizzly system. Fig. 2 shows drawpoint system from footwall. Fig. 3 drawpoint system from hanging wall. Brow of drawpoint has been left thick to conform to arc of bucket and strengthen drawpoint.

Eimco loaders will load cars full, handle larger pieces than could be permitted through grizzlies and eliminate spill (usually prevalent in chute loading).

Mechanical loading from drawpoints has been proved most economical by mining companies that have used both the mechanical loader and chutes and grizzlies. Some of their findings are as follows:

(a) Faster development, quicker production. (b) Less cost per installation. (c) Greater safety. (d) Elimination of chute maintenance and repair. (e) Fewer men required. (f) Flexibility of equipment permits operation in more than one drawpoint. (g) 100% salvage of installation when all ore is drawn.

Write for Bulletin L1017 on drawpoint loading with efficient Eimco Loaders.

EIMCO A430

THE EIMCO CORPORATION

The World's Largest Manufacturers of Underground Rock Loading Machines

EXECUTIVE OFFICES AND FACTORIES - SALT LAKE CITY 10, UTAH, U. S. A.

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IN ENGLAND: EIMCO (GREAT BRITAIN) LTD., LEEDS 12, ENGLAND
IN ITALY: EIMCO ITALIA, S.P.A., MILAN, ITALY

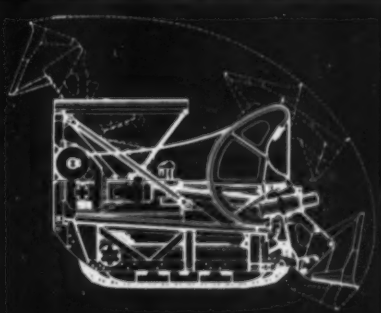
BIG LOAD
LITTLE EFFORT

Loads like this were considered jobs for big heavy equipment not long ago — but today loads this big or bigger can be handled easily with plenty of horsepower in reserve in the power unit.

The secret of the Eimco 104 is in better engineering, better steel castings and the rocker-arm principle described at right — where effort is applied to obtain a maximum in mechanical advantage.

Eimco 104's are heavy-duty loaders. They'll handle rock as easily as sand and gravel. Bucket sizes vary with material being loaded with an average job loading rate of between 200 and 300 yards per hour depending on job conditions.

Eimco 104's are being used for highway maintenance, heavy earthmoving, contracts, tunnels, steel mills, sand and gravel pits and many other jobs. Write for more information on Eimco loaders. Built in 6 different models for all loading jobs.



The rocker-arm principle is used on all Eimco Loaders. This principle, designed to give maximum force at the digging lip while the bucket is being worked up through the muck pile permits crowding forward at the same time so that the bucket will come up full every load.

EIMCO A431

You can't beat an Eimco

THE EIMCO CORPORATION

The World's Largest Manufacturers of Underground Rock Loading Machines

EXECUTIVE OFFICES AND FACTORIES - SALT LAKE CITY 10, UTAH, U. S. A.

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Send him into a stope or raise or crosscut.

Send him with a Gardner-Denver S48 Drill and FL2A Feed Leg Mounting.

The time he saves handling this lightweight equipment — plus the drilling pace set by the power-packed S48 — will enable him to do a man-sized drilling job every day. See Gardner-Denver Bulletin HHD-11 — or write for a copy today.

SINCE 1859

GARDNER-DENVER

Gardner-Denver Company, Quincy, Illinois
In Canada: Gardner-Denver Company (Canada), Ltd.,
14 Curity Avenue, Toronto 13, Ontario

THE QUALITY LEADER IN COMPRESSORS, PUMPS AND ROCK DRILLS

NEW TECHNIQUE OF COBALT FLOTATION was worked out in a pilot plant at Calera's Cobalt, Idaho installation to mill ore assaying 0.70 per cent cobalt. Reliability of the mill operation is promoted by the G-E engineered electrical system.

Calera will concentrate 1000 tons per day to ease cobalt shortage

Output kept continuous at new mine and mill by G-E engineered electrical layout

Soaring military demands for scarce cobalt—vital metal used in jet engines, armor plate, shells, and other weapons—will be met in 1953 with the help of the Calera Mining Company's new mine and mill at Cobalt, Idaho.

To keep the concentrator operating continuously, with fewest possible outages, this Howe Sound subsidiary turned to General Electric during the planning stage for help in engineering the electrical system. G-E application engineers designed—as a co-ordinated unit—the electrical layout for the mine, the mill, and the new town of Cobalt.

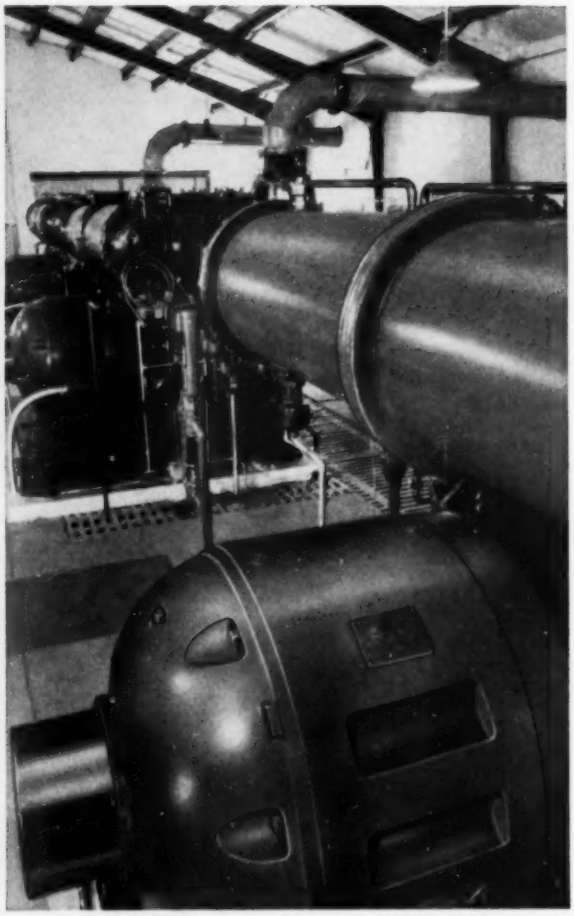
Thanks to the carefully planned flexibility of this electrical system, the Calera Company met government requests to increase the concentrator's capacity by 40 per cent without serious delay.

GENERAL ELECTRIC

TURN PAGE FOR PHOTO REPORT

1. ORE TRANSFERRED ON MINE LEVEL is handled by this 4-ton General Electric battery trammer.

2. DEPENDABLE VENTILATION—to clear the air after blasting in the mine—is provided by this 30-hp totally-enclosed G-E motor. It runs continuously when the mine is worked.

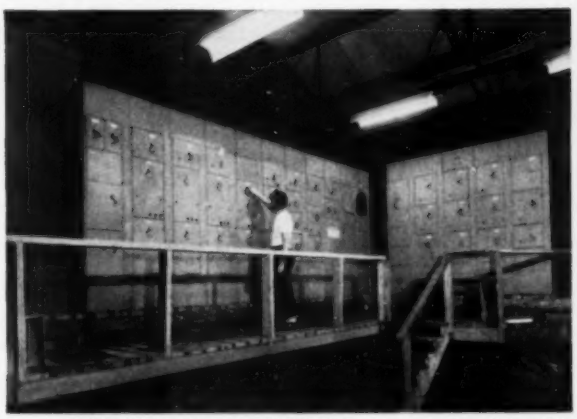


3. AIR COMPRESSORS are driven by 300-hp G-E motors. Compressor building also houses a standby diesel generator which supplies emergency power for surface plant facilities.

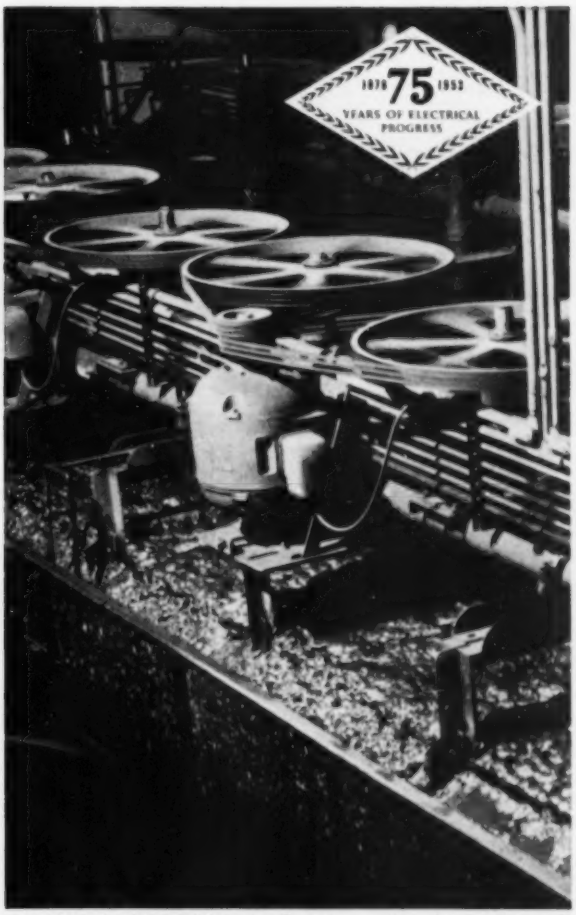
Major cobalt mine in U.S. relies

THIS 1000-TON CONCENTRATOR at Cobalt, Idaho, separates a copper, a cobalt, and an iron product. Crushing plant and concentrator are located close to mine portal. Treatment involves long conditioning of the pulp with lime at 80 to 90 degrees Fahrenheit.

4. PRIMARY AND SECONDARY CRUSHERS—driven by G-E Tri-Clad* motors—are supplied from a 480-volt wye-connected, solidly grounded G-E load-center unit substation.
*Registered trade-mark of General Electric Co.



5. CONTINUITY OF PRODUCTION is better assured by this General Electric motor control center. It contains small motor starters, feeders for crane, lighting and power circuits.



6. ORE IS PROCESSED in flotation cells, driven by G-E motors, then concentrates are shipped to the Howe Sound Company's new $2,500,000 cobalt refinery at Garfield, Utah.

on co-ordinated electrical system

More than two years of pilot-plant testing—and the development of a novel technique of selective cobalt flotation—was needed before the new Calera mill started operating as the first major source of domestic cobalt.

To assure reliable power distribution and motor operation for the new installation, Howe Sound specified G-E equipment throughout. G-E application engineers, working closely with Howe Sound's engineers, designed a completely co-ordinated electrical system. Further, G-E service engineers were on hand for equipment start-up and during the "breaking-in" period. As an indication of their satisfaction with the electrical system, Howe Sound specified G-E equipment for their new $2,500,000 cobalt refinery at Garfield, Utah, just now going into operation.

Whatever your electrical needs—large or small, modernization or expansion—G-E application engineers can help you and your consultants. Contact your local G-E Apparatus Sales Office. General Electric Co., Schenectady 5, N. Y. 660-28

Engineered Electrical Systems for the Mining Industry

GENERAL ELECTRIC

Meet The Authors

A. D. BRYANT

W. S. ROW

E. B. FITCH

A. D. Bryant (*Hydraulic Method for Underground Silica Sand Mining*, P. 282) is vice president of the Standard Silica Corp., Chicago, Ill. His career as a mining engineer began with graduation from Lehigh University in 1910. Mr. Bryant was employed by Bethlehem Steel Co. as a junior mining engineer in Cuba. From 1912 to 1916 he was assistant chief of the zinc oxide department of New Jersey Zinc Co., Palmerton, Pa. Positions with Grubnau, Bryant & Grubnau; California Zinc Co.; and Cerro de Pasco Copper Corp., preceded his association with Standard Silica in 1944. Mr. Bryant was a member of the wrestling team while at college. He resides in Ottawa, Ill., and is a member of the AIME.

W. S. Row (*Some Features of Current Mining Practices at Kerr-Addison Gold Mines, Ltd.*, P. 294) received his degree in mining from McGill University and went to work as an engineer for Cerro de Pasco Copper Corp., Peru, remaining there for three years. Before joining the staff of Lake Shore Gold Mines, Ltd., Mr. Row worked for Frood Mine. He came to Kerr-Addison Gold Mines, Ltd., in 1937 taking over as manager. A resident of Virginiatown, Ont., he fishes, hunts, and plays golf in addition to curling.

E. B. Fitch (*Operating Behavior of Liquid-Solid Cyclones*, P. 304) has diverse interests, which include little theatre activities, plant hybridizing, square dancing, and fencing. While at the California Institute of Technology he was a member of the varsity fencing team, dramatic club, glee club and various technical clubs. After receiving his degree in 1932, Mr. Fitch worked for American Potash & Chemical Corp., in charge of developing soda products. He has been with the Dorr Co., since 1944. He is now assistant research director.

L. J. Boucher (*Cement Rock Beneficiation at the Universal Atlas Cement Co., Northampton, Pa.*, P. 289) earned letters for three years as a member of the varsity football team at Missouri School of Mines. A member of Sigma Nu, Mr. Boucher also belonged to the Pan Hellenic Society while at school. He has been employed as a mining engineer by

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L. J. BOUCHER

L. A. ROE

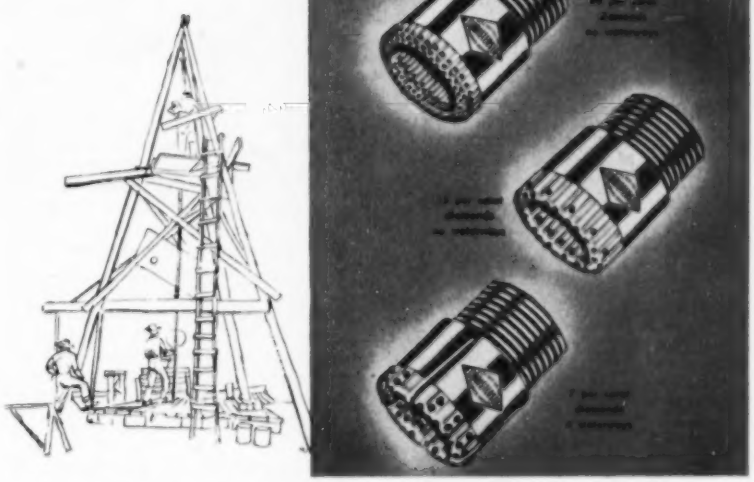
R. J. CORFIELD

Fentress Coal & Coke Co., Wilder, Tenn.; Arctic Coal Co., Island of Spitzbergen; Utah Copper Co.; and Atlas Portland Cement Co. He has written articles on quarrying for technical publications and is a member of the AIME. His principal interest, aside from business, is community service. He is active in school board work, boys' club, and boy scouts.

L. A. Roe (*The Magnetic Reflux Classifier*, P. 312) calls his profession his number one hobby. Mr. Roe earned a Bachelor of Science in Mining Engineering in 1939, after attending the Wisconsin Institute of Technology and the Missouri School of Mines and Metallurgy. After working for Nevada Consolidated Copper Corp., Mr. Roe held positions with American Potash & Chemical Corp., Battelle Memorial Inst., and Jones & Laughlin Steel Corp. which led to his present employment with the Bjorksten Research Laboratories as director of the mining and metallurgy section.

R. J. Corfield (*Electrical and Metallurgical Improvements at Kennecott's Utah Copper Mills*, P. 274) is currently at Kennecott's Garfield facilities, after employment by General Electric Co., Westinghouse Electric Corp., and Great Western Power Co. Mr. Corfield has also been a consultant for Six Companies in Las Vegas, Nev. He holds a Master's in Electricity and is a graduate of Eastern College. He has authored several published papers on electricity. Golf is his major interest, and he manages to trod the fairways at least three times a week, swinging his clubs left handed for a score in the low 90's.

E. C. Johnson (*Operating Behavior of Liquid-Solid Cyclones*, P. 304) has been a serious amateur camera fan since working as a professional photographer and assignment as a Naval cameraman aboard the unsteady decks of a destroyer during World War II. He has been employed by the Dorr Co., since graduation from Clarkson College of Technology with a Bachelor of Science in Chemical Engineering. While at school he was elected to membership in Omega Chi Epsilon. Mr. Johnson is a resident of Bridgeport, Conn.



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Books for Engineers

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Gasification and Liquefaction of Coal. *American Institute of Mining and Metallurgical Engineers.* $6.00 to nonmembers, $4.20 to members, 221 pp., 1953.—The volume contains the papers presented at the Symposium on the *Gasification and Liquefaction of Coal*, held at the Annual Meeting of the AIME, February 1952. It presents a comprehensive view of worldwide developments in many phases of coal gasification. New and current developments dealing with techniques, plant design, direct hydrogenation, chemicals output, underground gasification, and gas economics are reviewed.

Phosphoric Acid, Phosphates and Phosphatic Fertilizers, by William H. Waggaman. *Reinhold Publishing Corp.* $15.00, 683 pp., 1952.—The book is a comprehensive treatment of the chemistry, technology, and uses of phosphorous and its important compounds. It covers deposits and the mining and recovery of phosphate rock. Manufacturing methods covered include economic aspects and special applications such as phosphating of metals. Technological and chemical information is also presented. The appendix contains complete lists of U. S. patents pertaining to manufacture and use of phosphorous compounds. The book is intended for technical men engaged in the recovery and processing of phosphorous rock, and executives and workers in pertinent industries.

Western Resources Handbook. *Stanford Research Institute.* One primary subscription, $250, secondary subscription, $100. The handbook is a compilation of economic and technical data on the resources of the 11 western states. It is not confined to natural resources, but includes also the agricultural, manpower, industrial and other resources necessary to the western economy. Presentation is graphic, the Handbook being made up primarily of tables, charts, and maps, with a minimum of text. Looseleaf makeup allows periodic additions. All Government and private data available to the Institute are used in the preparation of the Handbook. The Institute plans to remedy certain gaps by developing data which has not existed in organized form. The Handbook will be built up, rather than published complete at one time. The necessary binder, indexing tabs, introductory material, and other necessities are included in the subscription.

The Engineer's Approach to the Economics of Production, by John Reid Dick. *Sir Isaac Pitman & Sons, Ltd.* 21s, 248 pp., 1952.—The book deals with the engineer's problems relating to capital investment and operating costs. It covers behavior of factors determining efficient production. The application of theories of economics is used to further the ideal situation of optimum efficiency with minimum costs.

Symposium on Consolidation Testing of Soils. *American Society for Testing Materials.* $1.75, 109 pp., 1952.—The eight papers contained in the book present testing techniques, the effect of certain variables on soil characteristics, reports on actual consolidation tests, and discussions of observed settlement of railway and highway structures. A section devoted to general discussion follows the group of papers.

Recent U. S. Patents

2,626,052 Air-mechanical flotation machine. (1/20/53) *Robert Carbonnier,* Madagascar.

2,626,139/140 Mining machine for coal. (1/20/53) *Marver & Coulson Ltd.,* Glasgow, Scotland.

2,626,854 Purification of cuprous oxide (1/27/53) *C. K. Williams Co.*

2,626,759 Jaw crusher (1/27/53) *Hiram L. Barber and George L. Sellars.*

2,626,895 Electrolytic production of iron for powder metallurgical purposes (1/27/53) *Fansteel Metallurgical Corp.*

2,626,757 Attrition mill (1/27/53) *Allis-Chalmers Mfg. Co.*

2,626,721 Unloading system for mines, car changer for cages. (1/27/53) *Armitage & Bertram Ltd.*

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U. S. Bureau of Mines Publications

RI 4927 Concentration tests on various base metal ores. 14 pp.

RI 4929 Refractory properties of Pacific Northwest chromites. 38 pp.

IC 7650 State compensatory provisions for occupational diseases. 125 pp.

IC 7651 The lime industry. 43 pp.

IC 7652 Roof bolting the Delaware Aqueduct. 9 pp.

IC 7653 Economies through roof bolting in an Indiana coal mine. 10 pp.

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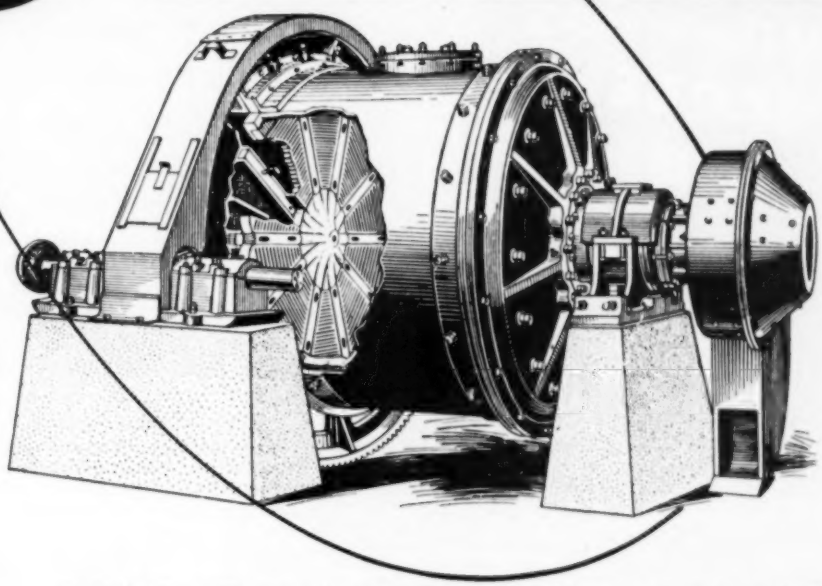

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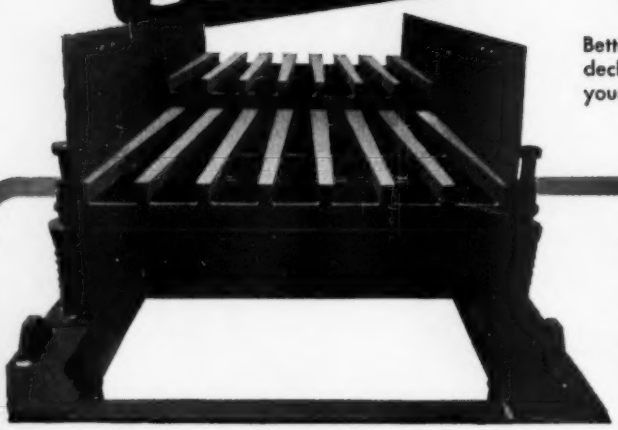

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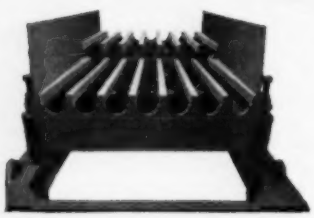
Better results in scalping with these special decks . . . designed to S-T-A-Y C-L-E-A-R on your heavy duty jobs.

Double Tapered Grizzly Bar Deck

(for openings 1 to 3 in.)

Longitudinal openings between grizzly bars stay clear because bars provide flared openings. Step deck turns material over, dumping fines from large pieces. Each bar is tapered from back to front and from top to bottom for additional freedom from plugging. Handles feed sizes up to 30 in. diameter.



Straight Grizzly Bar Deck

(for openings 4 to 10 in.)

For run-of-mine screen as shown above for use with straight instead of tapered grizzly bars. Openings between bars are flared from back to front for free material discharge. Handles feed in excess of 1000 tph in sizes up to 3 ft.

Steel Plate Deck with Skid Bars

(for openings 3 to 10 in.)

For run-of-mine scalping where positive scalping is desired. Steel plate decks with skid bars may be obtained with or without step deck construction. Plate deck is bolted to screen body and can be removed easily.

These special decks for primary scalping are available on both Style C and Extra Heavy Duty *Ripl-Flo* vibrating screens.

Ripl-Flo is an Allis-Chalmers trademark.

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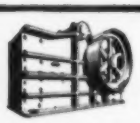
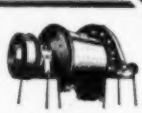


Jaw Crushers

Gyratory Crushers



Grinding Mills

Kilns, Coolers, Dryers

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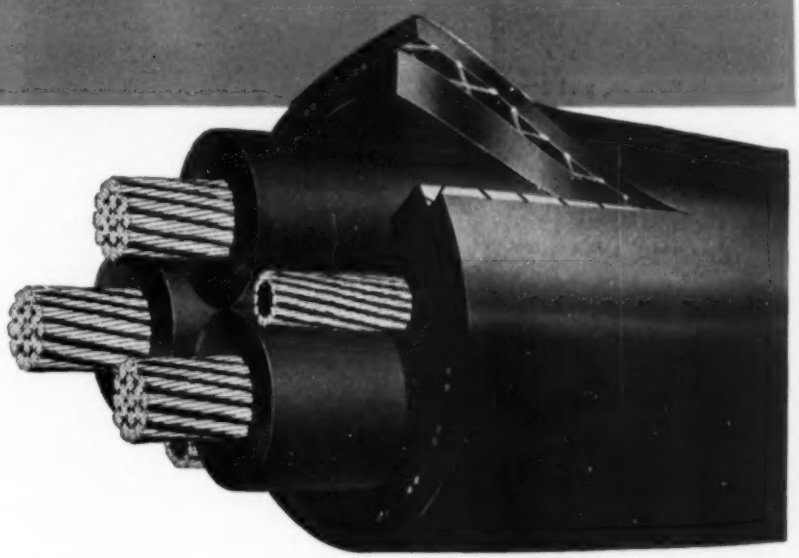

*Cold rubber is firmer and tougher and less permeable. It gives this new and better Type G Securityflex**

much more moisture resistance
much more compression resistance
much more cut resistance

That means longer cable life, fewer power interruptions, less maintenance trouble, *much more value for your money.*

Also – the famous Anaconda patented rubber-cored grounding wires are non-kinking and cushioned in a larger diameter that won't cut insulation.

This means far more dependable and efficient grounding. When the cable twists in service the grounding wires are protected. Every wire in every strand is firmly supported and cushioned. There'll be fewer failures, fewer splices, less production loss, less expense.

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WIRE AND CABLE

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Loader

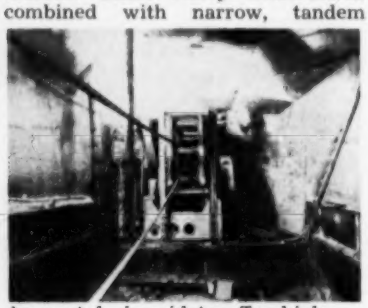
Built for trackless mining in low coal, the new *Goodman Mfg. Co.*

loader is claimed to load 8 tons per minute, has a head swing of 40° giving a 14½-ft loading path and good corner clean-up. Overall

height is 26½ in. and loading head swings 12 in. below mine bottom to 19 in. above. **Circle No. 1**

Slusher

Vulcan Iron Works Co. have developed a new Vulcan-Denver heavy duty slusher for large tonnage operation. Extra heavy construction combined with narrow, tandem



drum style is said to offer high capacity, mobility, and economy of space. A 150 hp motor provides 280 fpm rope speed, offering 1000-ton per hour capacity with 72-in. scraper. Four units have recently undergone trial runs at a large caving operation. **Circle No. 2**

Flexible Conduit

Sealtite flexible electrical conduit, type UA, is stated to be the first of its kind to receive Underwriters' Laboratory approval for use in wet locations. *American Brass Co.*, manufacturers, say that the extruded synthetic covering over the metal core protects against moisture, oil, chemicals and corrosive fumes. **Circle No. 3**

Emergency Light

Two 100 cp lights will provide instant light if power fails with *General Scientific Co.* emergency lighting unit. Storage battery to power lamps is kept charged by trickle charger. **Circle No. 4**

Drifter

Gardner-Denver Co. has introduced a 3½-in. drifter, model 93, called the "Eager Drifter" and claimed to equal most 4-in. machines in drilling speed, to exceed all other 3½-in. models in hole cleaning ability. Featured is the long feed aluminum alloy mounting with automatic feed motor on drill backhead or on guide shell. **Circle No. 5**

Scoopmobile

Two new 4-wheel drive, 4-wheel power steering, front end loaders in

1 and 2 cu yd sizes have been marketed by *Mixermobile Mfg.* Construction style utilizes a center pin steering coupling and two hinged power axle elements. **Circle No. 6**

Shot Core Drill

Designed to recover up to 20-in. cores from depths to 600 ft, the *Acker* model KR Shot Core Drill is said to represent an accumulation of over 25 years experience in design and development. Heavy duty rotary drill head has 3-speed transmission, cargo type hoist and positive displacement pump. Shot is fed through interior of drill rods to cutting bit. **Circle No. 7**

Coal Haulers

A fleet of 10 coal haulers of 47-ton capacity has been built by *Euclid Road Machinery Co.* for Hanna Coal Co. Largest ever built by Euclid,

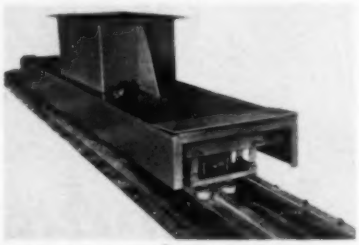
tractors are powered by 350 hp diesels and trailers are built of high alloy steel. A 10-speed transmission, air assist clutch and hydraulic booster steering aid smooth operation. **Circle No. 8**

Technical Data

Listing over 2000 subjects the *Lefax* general catalog of pocket size technical data books is now available. Data books contain 140 pages, sell for $1.00 each. **Circle No. 9**

Zipper Seal

Adapted from their zipper conveyor belt, *Stephens-Adamson Mfg. Co.* now offers a zipper bunker seal,



designed to prevent dust from traveling tripper discharge spouts feeding bunkers. **Circle No. 10**

Mine Jumbo

LeRoi-Cleveland self leveling jumbo is stated to speed up drilling cycles, cut costs. Air-motor-powered arm lets drillers spot holes quickly, without tilting or adjusting a boom. Offset arm provides clearance to drill lifters. Jumbo is built in single or double-arm construction. **Circle No. 11**

Screen

New style vibrating screen has been developed by *Hewitt-Robins Inc.* to handle heavier loads than previous models. The Robins Vibrex, model MS, employs the circle-throw principle, has heavy coil springs encased in neoprene boots, and will be available in single and double deck models. **Circle No. 12**

Free Literature

(13) ENGINEERING: *Western-Knapp Engineering Co.* has issued a well illustrated brochure describing organization and field offices, and explaining their integrated engineering service. Under the slogan "All . . . or part" they describe a project management service operating from "inquiry to turnkey."

(14) CRANES & SHOVELS: Eight articles discussing "Use and Application of Power Cranes and Shovels" have been compiled and distributed by *Koehring Co.* The 32-page booklet contains articles on size selection, and graphs for cost estimation.

(15) BOTTOM DUMP SKIP: *Lake Shore Engineering Co.* has available data on their Jeto bottom dump skip; claimed to be easier on headframe in dumping cycle, raise tonnage by faster dumping and clean more thoroughly when handling sticky material.

REDUCING DUST NUISANCE at quarry of Kingston Trap Rock Co. this two-piece, telescoping rubber hose supplied by Thermoid Co. helps bridge varying distance from conveyor discharge to top of surge pile. Cables attached to the flange and controlled from the hopper, raise or lower hose as surge pile height varies.

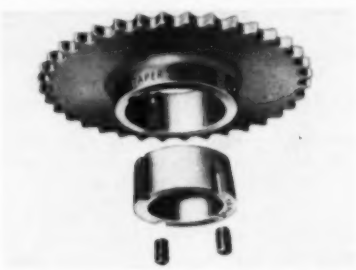
(16) TRAMP IRON: Team of iron detector, and magnetic pulleys to offer full crusher protection is described in catalogs from *Dings Magnetic Separator Co.*

(17) SCREEN CLOTH: *Hewitt-Robins* screen cloth in eight weaves, four types of wire, or of special material, is described in a new bulletin.

(18) GEOPHYSICS: Literature from *Georator Corp.* describes their complete Geovolt apparatus for self-potential and spontaneous polarization prospecting.

(19) CRUSHERS: *Denver Equipment Co.* has issued a bulletin describing their jaw crushers available from laboratory models up to 32x40-in. size.

(20) TAPER-LOCK: *Dodge Mfg. Co.* has introduced a new line of chain sprockets which use the Taper-Lock principle already proved in pulley service. New bulletin describes advantages of off-the-shelf applicability that eliminates boring of sprockets and turning and grinding of shafts.



(21) COMPRESSORS: Midget Angle, gas engine driven model HMA compressors in models from 80 to 330 hp are described in a new catalog from *Clark Bros. Co.* Primarily designed for petroleum industry they are stated to be suited to heavy construction projects using bottled propane as fuel.

(22) DIESEL PRIMER: Unique booklet, "What You Should Know About Diesel Engines" published by P&H Diesel Div., *Harnischfeger Corp.*, outlines 2-cycle and 4-cycle operation, explains engine principles.

(23) DRILL BITS: New *Kennametal* catalog illustrates their tungsten-carbide tipped drill bits for rotary and percussion drills, core and drag bits, augers, and pinning rods. Insert booklet shows comparative cost records from carbide bit users.

(24) LINK-BELT: A 46-page directory of products, and applications from *Link-Belt Co.*, also lists all offices and sales personnel in the U. S. and abroad.

(25) SHOVEL: The *Marion Power Shovel Co.* have described their model 111-M, 4-cu yd, diesel shovel in a new 12-page booklet.

(26) CONVEYORS: Spiral conveyors are the subject of *Jeffrey Mfg. Co.* catalog 851. Capacities, dimensions, drives and installations are covered in pictures and tables.

(27) EXPLOSIVES: "Explosives and Blasting Supplies," and "Hercomite and Gelamite Explosives for Lower Blasting Costs," are new booklets issued by *Hercules Powder Co.*

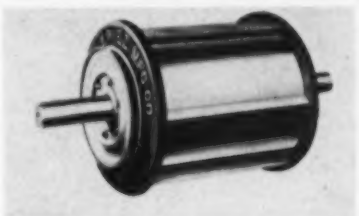
(28) TRANSFORMERS: *General Electric Co.* announced a 1953 Instrument Transformer Buying Guide containing ratings, ASA accuracy classification and prices of the complete indoor and outdoor line.

(29) TUNGSTEN-CARBIDE TOOLS: Latest information on when and how to sharpen tungsten-carbide tools is included in a new catalog and maintenance manual from *G.E.'s* Carboloy Department.

(30) TOURNADOZERS: Bulletin from *LeTorneau* describes construction and design of Tournadozers. The 186 hp engine provides speeds up to 19 mph on 18.00x25 low pressure tires. Transmission permits speed change under load without loss of momentum.

(31) WIRE ROPE: "The Right Rope for the Job," a wire rope recommendation book from *American Steel & Wire Co.* tells its story with photographs and diagrams.

(32) MAGNETIC DRUMS: Improvements and modification to increase efficiency have been made on *Eriez Mfg. Co.* permanent magnet drums



according to a new bulletin. Among other changes the Alnico V magnet is now enclosed in a stainless steel shell to resist pitting and corrosion.

(33) BUILDINGS: Steeline buildings in 32 to 70 ft widths and lengths in multiples of 20 ft are described in a bulletin from *Soule Steel Co.*

(34) MINE LOCOMOTIVES: *Goodman Mfg. Co.* recently issued a catalog for their complete line of trolley, trolley and reel, storage battery, and combination type mine locomotives.

(35) WELDING CAST IRON: Machinable welds on cast iron, made with Niclel-Arc 60 electrodes, are illustrated in a booklet from *Alloy Rods Co.*

Mining Engineering **March**
29 West 39th St.
New York 18, N. Y.

Please send me { More Information ☐ / Free Literature ☐ / Price Data ☐ } on items indicated.

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122, 186, 275-325, 450 h.p. . . . speeds from 28 to 40 m.p.h. Power steer . . . 90° turns . . . power-proportioning differential . . . multiple-disc air brakes . . . constant-mesh or sliding-gear transmission . . . torque converter optional on 186 to 450 h.p. units.

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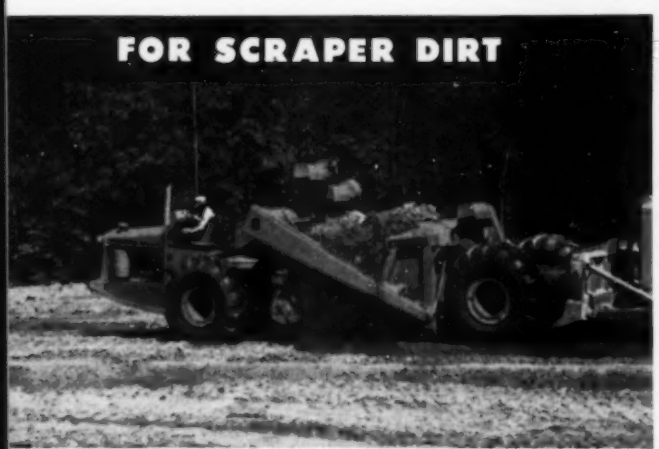

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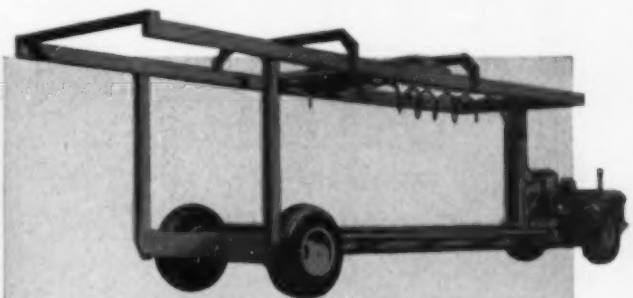
With scraper coupled to your prime-mover, you can strip overburden, move spoil banks, build and surface haul roads. Big low-pressure tires assure ample flotation in mud, sand and snow. Constant-pull, power-proportioning differential also reduces weather delays . . . gives positive traction on soft banks and slick pit grades . . . keeps your prime-mover working and earning all year 'round. Change from one hauled unit to another takes only a few hours. All units electrically controlled . . . use same wheels, tires, and brakes.



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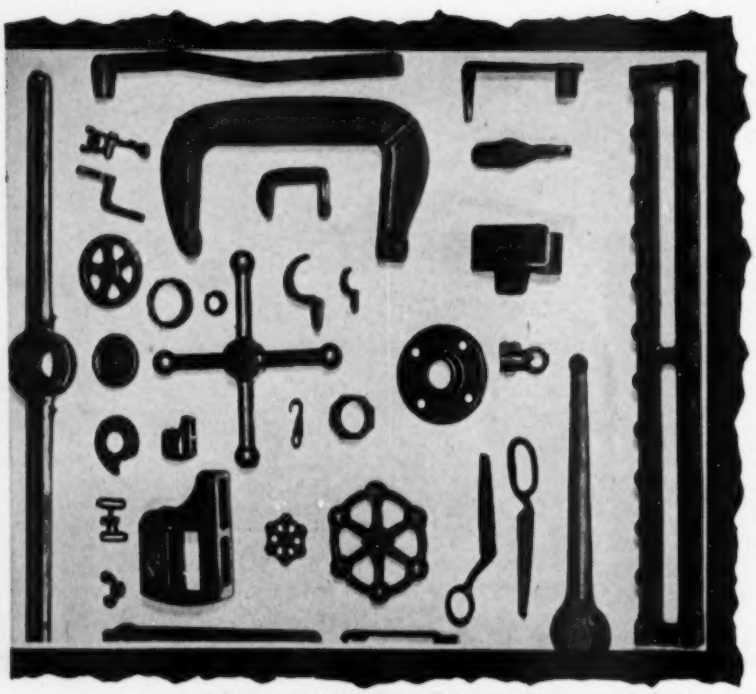
2. It can provide a chilled, carbidic, abrasion-resistant surface supported by a tough ductile core.

3. As-cast ductile iron of 93,000 psi tensile strength has the same machinability rating as gray iron with a strength of 45,000 psi.



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4. Annealed ductile iron can be machined at a rate 2 to 3 times that of good quality gray iron.

5. It can be satisfactorily welded.

6. It resists oxidation and growth to an extent never before available in gray iron castings.

7. It can be cast in intricate shapes not normally feasible for cast steel.

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Ductile Iron offers excellent castability, high mechanical properties and good machinability. Castings show superior pressure tightness, good elastic modulus and resistance to shock. They range from those weighing a few ounces . . . with sections as thin as one-tenth of an inch . . . to 50-ton anvil blocks with sections 4' thick.

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Objective stockpile goals have been met for antimony, bismuth, fluorspar, and iridium, while those for bauxite, cadmium, copper, lead, zinc, manganese, mercury, platinum, tin, and vanadium have not, according to a report by the Munitions Board. Total materials under inventory and contract represent about 78 pct of entire stockpile goals, the report stated.

Frobisher, Ltd., of Canada has been given permission to survey power possibilities in certain lake and river basins in the Yukon believed to have a potential of 5 million hp. Canadian Resources Minister, Robert Winters, also announced that his Government has rejected a bid by Aluminum Co. of America concerning use of Canadian waters to power a proposed Alaska aluminum project. Alcoa is reported hopeful of resuming negotiations.

Inland Steel Co. is reported to be negotiating an agreement with Steep Rock Iron Mines in northwestern Ontario to bring Inland into Canada "in a big way." Inland officials hope to announce signing of papers soon. Bethlehem Steel Corp., through its Canadian company, Marmoraton Mining Co., Ltd., has obtained patents for mining rights over new properties in the area east of Marmora, Ont.

Output of recoverable gold by U. S. mines declined 5 pct in 1952, the Bureau of Mines reported, while silver production dropped 2 pct. South Dakota led gold production with 484,556 fine oz. Idaho led in silver with 14,746,329 fine oz.

Anaconda Copper Co. has purchased mineral properties and equipment of the North Butte Mining Co. Total production of copper, silver, and gold since the mine started in 1905 has been $85 million from six shafts and 50 miles of underground workings.

Rep. Thomas E. Martin (R.-Iowa) has introduced a bill providing for Government purchase and stockpiling of domestic manganese ore or concentrates containing 15 pct or more of metallic manganese per long ton at a base rate of $1.00 per unit, with a sliding scale price as metallic content increases.

First 10,000-ton shipment of bauxite was loaded for shipment to the U. S. at Kaiser Aluminum & Chemical Corp.'s bauxite mining operation in Jamaica. The properties are expected to have an annual capacity of about 2 million tons.

Suspension of proposed reduced railroad rates on imported iron ore from Boston, Baltimore, Philadelphia and New York to Youngstown, Pittsburgh, Wheeling, and nearby areas, has been announced by the Interstate Commerce Commission. Suspension will be in effect until September 8 pending investigation.

U. S. Army engineers have approved plans for dredging the Delaware river to facilitate ore delivery to the Fairless Works. Study by various Government departments is in progress.

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JOY TELESCOPIC FEED STOPERS

The S-91T provides a steel change of 36", compared with the 18" change of conventional stopers. In most mining conditions, this long feed will give more drilling time, with fewer steel changes and fewer lengths to stock.

At left, the Joy S-91T Stoper in operation; above, with telescopic feed leg retracted; right, with leg extended, giving a 36" drilling feed. Another feature, instant; "thumb-flip" rotation release, provides safety and easy spotting of holes.

JOY HAND-HELD DRILLS

A complete line of well-balanced, compact drills, ranging from 27 to 67 lbs. in weight.

The L-57 and L-67 Joy Hand-Held Drills are also available with mounting brackets for use as light drifters, or for wagon drill mounting.

Joy Jibs are hydraulically controlled booms for drill mounting on Jumbos, or on rubber-tired, self-propelled Drillmobiles.

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(1) The exclusive DUAL VALVE, a feature of all Joy rock drills, makes air do more work. By admitting the correct amount of air behind the piston and excluding air ahead of it during the drilling stroke, maximum force is exerted on the drill steel. There's no cushion of air ahead of the piston to reduce the force of the blow. Then the rear section of the Dual Valve meters the correct amount of air ahead of the piston to force it back with a "snappy" rotating action. This adds up to more power-packed strokes per minute, for faster drilling with less air.

(2) The silver-like cadmium plating on Joy *Silver Streak* Rock Drills aids lubrication while running-in, protects parts from rust while in stock, and keeps maintenance costs at a minimum.

The Joy line is complete. There's a *Silver Streak* rock drill for every rock breaking need. Shown here are drifters, stopers, hand-held drills, Hydro Drill Jibs. To complete the line, there are wagon drills; the QB-20 for line drilling and dimension stone quarries; rubber-tired, self-propelled Drillmobiles; and specially mounted drills for special needs.

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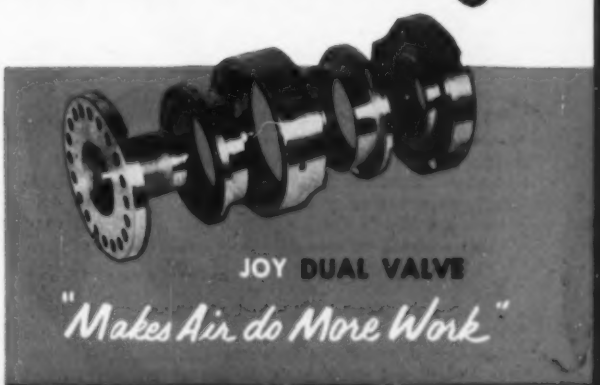

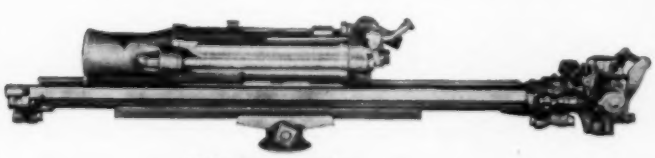

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This rugged, Chicago Pneumatic Diamond Drill has delicate bit control which keeps bit costs down and affords high core recovery. Has self-aligning rod puller for holes deeper than 100 feet. Conservatively rated at 500 feet with E Rods and EX Fittings. For more details, write for Bulletin 318. *Chicago Pneumatic Tool Company, 8 East 44th St., New York 17, N. Y.*

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Expect Labrador Iron Ore Shipments by 1954

Iron ore shipments from the Quebec-Labrador area are a "definite expectation" by late summer, 1954, according to William H. Durrell, general manager of the Iron Ore Company of Canada.

About $120 million have been invested in the project, with a total of $200 million expected to be reached before a ton of ore is shipped, Mr. Durrell said. According to the general manager approximately 418 million tons of open pit highgrade ore have been proven in the project.

Deposits lie several hundred miles north of the mouth of the St. Lawrence River in territory which was without transportation of any kind. Construction of a railroad from Seven Islands to Knob Lake, about 360 miles, is an important part of the development plan. The railroad will be the first to adopt complete roller bearing applications. Construction is on schedule with 215 miles of the route graded. Completion involves two tunnels through rock, one of almost half mile length, a 700-ft bridge and several smaller spans.

Building plans include two hydroelectric plants, loading docks, freight terminals, office buildings and housing. Hollinger Consolidated Gold Mines Ltd., of Canada, in association with the M. A. Hanna Co. of Cleveland started the project. Later, they were joined by five U. S. steel producers—Republic, National, Armco, Wheeling, and Youngstown Sheet and Tube.

Inco Installs 20-ton Ventilator at Creighton

A 20-ton exhaust fan has been installed by International Nickel Co. of Canada to ventilate underground workings of the new caving project at its Creighton Mine in the Sudbury District of Ontario.

The giant vertical fan, driven by a 350 hp motor weighing five tons, ventilates the workings with a flow of fresh air at the rate of 300,000 cfm. Air is driven directly through caved or broken ore, circulates underground, and is carried back to the surface through a main return shaft.

The fan rises 42 ft from its concrete base, has 15 ft diam at the inlet. The 124-in. impeller has 12 stainless steel blades which are adjustable in pitch through 25° to accommodate changes in load as mining operations move farther away from the return air shaft.

Hanna Subsidiaries in U. S. Nickel Project

One of the few major U. S. nickel deposits will be exploited jointly by the Hanna Coal & Ore Co. and the Hanna Nickel Smelting Co., subsidiaries of the M. A. Hanna Co. of Cleveland.

The two companies will exploit the deposit near the summit of Nickel Mountain in southwestern Oregon under terms of a contract signed with the Government. The companies will develop the mine and construct crushing and drying facilities and a multi-furnace smelter. Final product will be ferro-nickel ingots containing between 25 and 75 pct nickel.

Ferro-nickel can be used as a substitute for pure nickel in stainless steel and other low alloy steels. Defense Material Procurement Agency agreement with the Hanna Coal & Ore Co. provides that the company will develop the mine at its own expense. The firm will sell the Government enough ore to produce a minimum of 95 million lb of nickel contained in ferro-nickel. Mine development is expected to cost $4.3 million.

The Government will pay $6 per dry ton for ore with a nickel content of 1.5 pct—average grade indicated by test drilling. DMPA will advance Hanna Smelting up to $24.8 million for construction of a smelter and related expenditures. The company will convert ore to ferro-nickel ingots.

Price for the first 5 million lb will be not more than 79.39¢ a lb, and 60.5¢ per lb thereafter, exclusive of amortization of the facilities. The mine is scheduled to begin operation during the summer of 1954, and the smelter the following October.

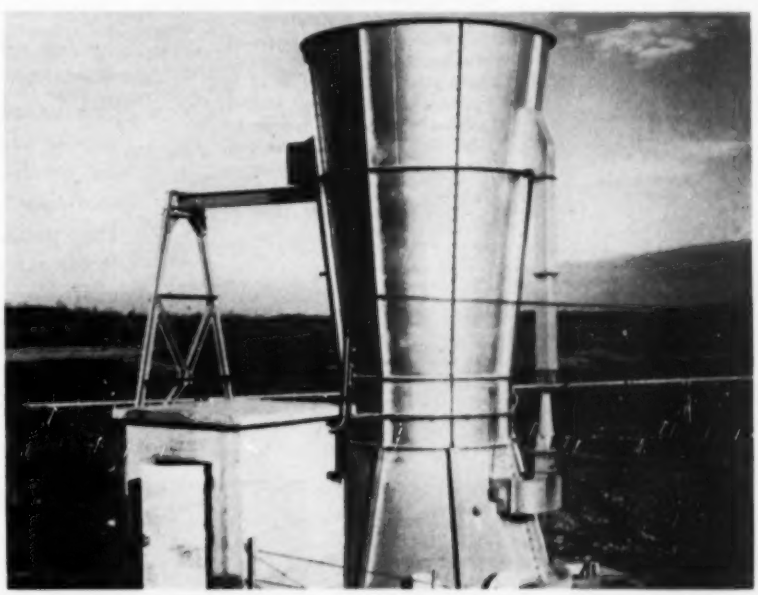
Pennsylvania Coal Buys Bituminous Operation

Pennsylvania Coal & Coke Corp. has purchased a bituminous coal mine in central Pennsylvania with an annual productive capacity of about 400,000 tons.

L. D. Silberstein, chairman of Pennsylvania Coal, said the new mine, together with increased production planned at other mines, is expected to boost sales in 1953 to about 1.8 million tons, or 50 pct more than the 1.2 million tons last year.

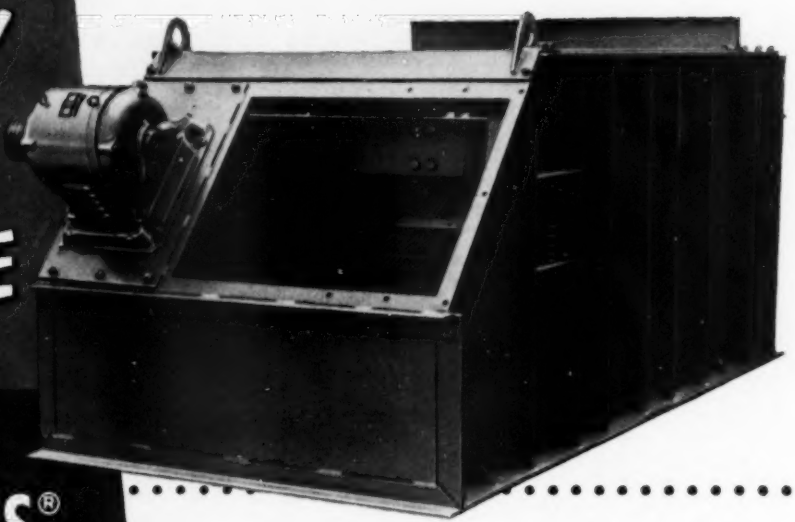
Under an expanding mechanization program, Pennsylvania Coal has purchased six additional continuous miners and auxiliary shuttle cars from Joy Mfg. Co.

The Pennsylvania chairman also said the new acquisition is expected to supply sufficient revenue from resulting increase in production to more than make up for reduction in profit margins brought about by wage boosts.



The giant fan ventilating International Nickel Co.'s Creighton Mine in the Sudbury District, Ont., stands 42 ft above its base and delivers 300,000 cfm of fresh air. Fan is mounted on top of main return shaft. Air is drawn down through broken ore and is returned through shaft.

FOR HEAVY DUTY SERVICE



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THE Symons Type K - RG vibrating Rod Grizzly is built for heavy duty service in handling large tonnages of dry, wet, sticky or gummy ores and rock. Here are some of the important features that have been incorporated into the Symons Rod Grizzly:

- *Heavy duty construction throughout . . . built to handle large tonnages under severe operating conditions.*
- *Abrasion resistant spring steel rod screening surface.*
- *Long lasting, low cost rods are easily adjusted or replaced.*
- *Effective, vigorous vibration provides thorough scalping. Amplitude of vibration can be quickly and easily changed to meet various operating conditions.*

This grizzly is ideally suited for service following most sizes of primary crushers and scalping operations requiring separations at ¾" or larger.

The Symons Bar Grizzly is recommended for large tonnage scalping where clear spacings of 1½" and larger are permissible. The powerful vibrating action of the Bar Grizzly makes it particularly effective for wet, sticky or gummy materials. The sloping, curved bars of the Symons Bar Grizzly tend to tumble the material as it passes over the screening surface. The tumbling action effectively assists in separating the undersize.

For full details about the SYMONS ROD GRIZZLY, send for your copy of BULLETIN 207.

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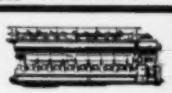

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Cleveland Belt Conveyor Plan May Solve River Jam

Would Ease Ore Traffic to Mills

Cleveland, Ohio has been in the position of a man trying to run with his feet in a potato sack. Its steel industry desperately needs room for expansion but is shackled to the narrow, torturously curved Cuyahoga River for raw material delivery. Cleveland's industry stops where the river becomes unnavigable, and so the flatlands of the upper Cuyahoga Valley have been going to waste. The answer may lie in a proposed rubber conveyor belt.

If constructed the conveyor would eliminate the necessity of using the river. Ore, limestone and other materials could be unloaded at new waterfront facilities on Lake Erie. Cargos could be stockpiled on the docks or placed directly on the belt for transport to steel mills. Spur lines off the main belt would carry material to individual user stockpiles.

Right now, river traffic is a perennial multi-million dollar problem to Cleveland. Engineers of the B. F. Goodrich Co., Akron, Ohio studied the conveyor idea for more than a year. They estimate it could be constructed at a cost of about $6 million for the conveyor and drive equipment. Dock and loading facilities, as well as rights of way, would be additional. Some city officials say the overall cost would be approximately $10 million.

The plan is currently under consideration by Cleveland's Port and Harbor Commission. Mayor Thomas Burke has urged industry and civic leaders to give the plan serious thought. James Lister, city planning commissioner, feels that the plan is well worth exploring even in the face of legal and financial problems which usually arise when projects cut across private and public property. The job of widening and straightening the Cuyahoga River is estimated at between $20 million and $40 million.

Flexibility of conveyor belt system is highlighted by an artist's conception. The road can go anywhere, up or down hill. Ore, in the proposed system for Cleveland, would be unloaded at dockside directly to belt.

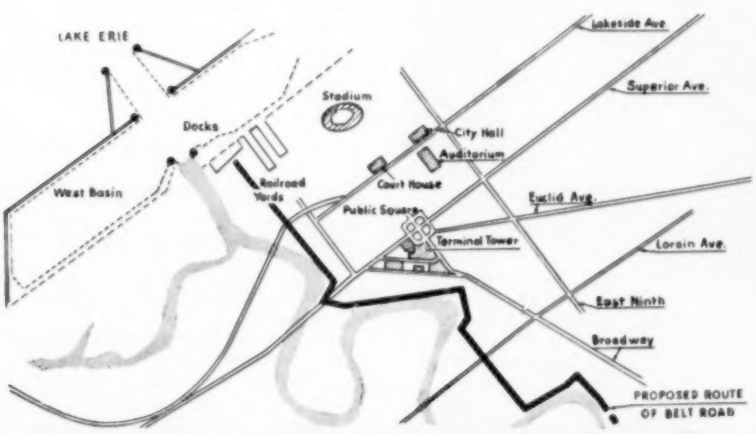
The artist's conception illustrates a 90° direction change as the belt follows the Cuyahoga River up the valley. Terrain in the valley is considered ideal for the system. Plans include a spur line system.

Transportation Block Prevents Mill Growth

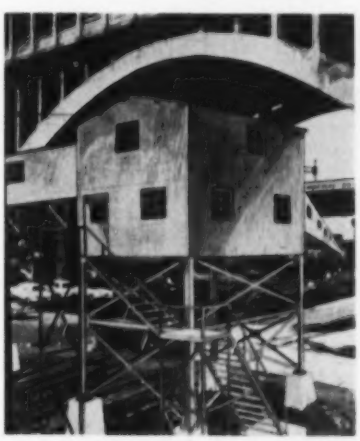
Ore boats are creating a definite bottleneck. River passage is slow and vessels could be put to increased use if they operated solely on the lakes. Use of a belt would in effect multiply the size of the ore fleet by making more trips during the season possible. Larger boats could be used. The narrow Cuyahoga will not permit passage of larger craft.

Cuyahoga Valley terrain is ideal for a conveyor system, posing no engineering problems, according to a rubber company report. Direct advantages would be felt by Cleveland's population, who often are forced to sit out block long traffic tie-ups as drawbridges are raised to permit ore boat passage. Cleveland's smoke problem would also be lessened somewhat by elimination of ore boats from the river. In the event of damage by air attack repair would take hours or at most days, as against weeks to clear the river for traffic.

Conveyors are growing in other areas, with Chicago's Museum of Science and Industry showing a moving sidewalk. Museum officials visualize a future where conveyor belts will play a major role in making city life a little more bearable—and incidentally a lot safer. Conveyors can be used to move people relatively short distances wherever traffic is heavy.



A possible route for the belt road has been mapped out. The road could be elevated to gain clearance for street or railroad traffic. Rights of way from industrial occupants of proposed route will be necessary. Broken line indicates where road begins to pass through mill property.



Drawing depicts appearance of the belt when passing under Detroit-Superior Bridge enroute through the valley. Proponents of the belt say it will be able to carry all ore Cleveland industry may need.

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Cleveland-Cliffs Plans Lowgrade Ore Project

Cleveland-Cliffs Iron Co. directors have approved a second project aimed at development of lowgrade Michigan iron ores. The firm's latest project will be at its Republic mine.

While no cost figures have been given for mining and concentrating the hematitic jasper, Cleveland-Cliffs has been granted a certificate to write-off 75 pct of $26,575,700 for the project at the fast rate. Early in 1951 Cleveland-Cliffs and Ford Motor Co. joined in an operation at Humboldt, Mich. Production is expected to start early in 1954 at the first project.

The Republic operation is located in the southwest part of the Marquette Iron Range. The process to be used in the new Republic open pit and concentrating mill is expected to produce an annual 400,000 tons of concentrates. Concentrator construction is expected to start this year.

A rail line was recently extended into the valley by the Lake Superior & Ishpeming Railroad to ease entry of construction equipment and materials to build the mills. Two concentrating units are planned, each capable of producing 200,000 tons of concentrates annually. Production is predicted for mid-1955.

Broken ore will be delivered to secondary crushers and grinding machinery before flotation treatment.

Zinc Price Curtails Two Idaho Operations

Nosediving zinc prices caused curtailment of operations in two Idaho mines, with accompanying cut in employment and limitation of activity.

Bunker Hill & Sullivan Mining & Concentrating Co., and Day Mines, Inc., both in the Coeur d'Alene district of northern Idaho were the companies involved. One of the West's largest lead and zinc producers, Bunker Hill laid off 130 of its 1400 employees. J. B. Hafner, Bunker general manager, said the firm has suspended work on block caving of lowgrade ores and on tailing reclaiming projects. Company officials said other employees would be shifted to higher grade ores and overall production would not be affected.

Henry L. Day, president and general manager of Day Mines, said production has halted from two marginal zinc stopes in the Tamarack mine near Wallace. Several men have been laid off. Other operations are unchanged. Most of the firm's production is lead.

Magma Copper Buys St. Anthony Property

St. Anthony Mining & Development Co. has received 10,000 shares of Magma Copper Co. capital stock in exchange for substantially all of its properties at Tiger, Ariz.

St. Anthony halted business and mining operations in 1952. It has been liquidating and disposing of property since that time. St. Anthony property adjoins those of San Manuel Copper Corp., wholly owned subsidiary of Magma Copper, where a large lowgrade copper ore development is in progress.

Primary purpose in acquiring St. Anthony Mining Co. is to obtain that firm's housing, dispensary, equipment, warehouse, schoolhouse, store, buildings and other structures. It gives San Manuel immediate housing and other facilities for carrying on its work.

St. Anthony underground workings also contain a source of water. St. Anthony is understood to be retaining its shares as an investment and will not place them for resale.

Stonega Coke Begins $3 Million Expansion

A $3 million expansion program is being launched by Stonega Coke & Coal Co., Big Stone Gap, Va., with a new colliery, a completely mechanized mine, and a new preparation plant included in the plans.

The colliery will be located at Crossbrook, north of Appalachia. The new mine is within one and a quarter miles of the nearest railroad. Coal will be carried to shuttle cars and then to conveyor belts for the final trip to the preparation plant. Stonega says it will be able to mine about 2500 tons of coal daily for the next 20 years.

The new preparation plant at Stonega Pine Branch Colliery will include a heavy density washer, new bottom-dump mine cars and a conveyor and bin system expected to eliminate tipple stoppages.

Soft Coal Demand Seen Rising in 1953

Coal requirements of several major industries are expected to place 1953 demand for soft coal about 10 million tons over the 1952 output of approximately 465 million tons, according to preliminary estimates of the Defense Solid Fuels Administration.

Electric power utilities, coke ovens, and steel mills are expected to increase demands as a result of defense program expansions. Export and railroad demands may decline in 1953,

but overall soft coal requirements are expected to reach about 475 million tons. Bituminous coal production may decline below this level, since industrial users may draw upon relatively large stockpiles.

In the meantime, the Coal Exporters Association reports that coal exports from the U. S. to all nations except Canada declined more than 25 pct in 1952 from previous year totals. Overseas shipments of both bituminous and anthracite coal in 1952 came to 24,682,000 gross tons, compared with 33,161,000 in 1951. Last year's total, including shipments to Canada, came to 46,145,000 tons, compared with 56,313,000 tons in the previous year.

Silver King, Park-Utah Plan Consolidation

Directors of Silver King Coalition Mines Co., and Park-Utah Consolidated Mines Co., meeting independently, have approved a plan to merge the two firms. Proposals were placed in the hands of lawyers to draw a formal agreement.

In a joint statement, James Ivers, president of Silver King, and Lawrence Fox, president of Park-Utah, said each board had agreed tentatively that consolidation would be achieved by forming a new corporation capitalized at six million shares of $1 par value stock.

Properties of Silver King and Park-Utah are located in the Salt Lake City area and are adjacent. Both produce lead, zinc, copper, gold, and silver. They have been closed in recent months because of low lead and zinc prices.

In 1950 Park-Utah and Silver King began joint development of an area bordering their mines. Officials of both firms agreed that consolidation will lead to greater efficiency in exploration, production, drainage and similar mining operations.

American Smelting and Refining Co. owns 303,369 shares of Silver King, while Anaconda Copper Mining Co. owns about 478,125 shares of Park-Utah.

Utah Construction Co. Ships Equipment to Peru

Utah Construction Co. began shipping heavy equipment to its Peruvian iron ore mining project. Equipment is consigned to the Marcona Mining Co., an American corporation in which Utah has an interest.

Included in the manifest were two Manitowoc shovels, steel girders, trucks, bulldozers, and lumber. Several railroad cars were needed to bring the shovels to San Francisco.

When mining begins ore will be shipped to Philadelphia, Baltimore, and Mobile.

Improved Magnetic SEPARATOR for CONCENTRATION and BENEFICIATION of

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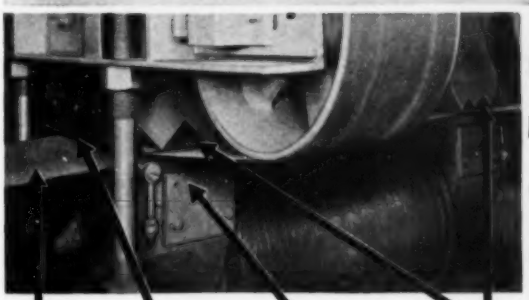
and SIMILAR MINERALS

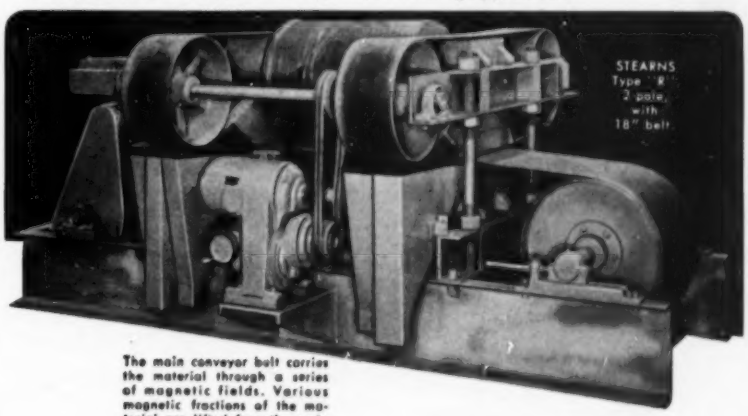


Stearns CROSS-BELT SEPARATOR

Utilizing the time-tested lift method of magnetic separation, the STEARNS Type "R" Separator has been re-designed to provide additional and improved separations of magnetically reluctant ores and minerals at a lower total cost. These results have been achieved mainly through a multiple point adjustable pole construction* which will give up to four different and distinct magnetic products at each pole. Thus, fewer magnet assemblies are required and therefore the initial and operating costs are reduced.

Having multiple magnetizing zones of increasing intensity in a single field not only provides (1) the time necessary to overcome the magnetic reluctance, but (2) also provides for a self-induced magnetizing effect on the ores and minerals as they pass from zone to zone of increasing magnetic intensity.

By adjusting the angle of the lower pole it is possible in most applications to use a single magnet separator having as many or more separating zones as would ordinarily be found in a separator having a multiple of magnetic fields. Write for complete descriptive literature on STEARNS Separators for mining applications.



The main conveyor belt carries the material through a series of magnetic fields. Various magnetic fractions of the material are lifted from the main belt and carried away by the cross belt assemblies.

*Patent No. 2511484

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THAT raw north country, Alaska, may someday have a steel industry of its own. Five years ago there seemed little need of this kind of industrial development, but since then population has increased by 77.4 pct. Railroads, highways, and bridges are slowly drawing the territory into a cohesive unit. Today, steel must be brought to Alaska from the United States. Along with growing civilian needs for steel and steel products are the demands of large and permanent military installations which have made Alaska an outpost of democracy.

Until 1950 the idea of a home steel industry was given little or no thought by Alaskans or anyone else for that matter. Alaska, source of some of the greatest history making events in mining lore, has almost totally lacked recent prospecting. The voluminous *Alaska Reconnaissance* offers this as a reason for the fact information on two iron ore deposits in the country has been slow.

Largest of the two deposits is at Klukwan Village 30 miles from Haines on Skagway Fiord, a deep, tidewater channel. The orebody is a vertical mountainside rising from an altitude of 400 ft to nearly 5000 ft. A good highway connects the deposit with Haines. Preliminary examination by the Bureau of Mines in 1951 estimated the deposit to contain from one to five billion tons of 15 to 20 pct iron ore as disseminated grains and small lenses of magnetite containing about 2½ pct titanium.

A Seattle broker, C. T. Takahashi, has the property under lease for a Japanese steel firm and has done some preliminary exploration work. A zone of relatively highgrade ore was discovered within the main deposit. Preliminary assays range from 30 to 50 pct magnetic iron.

Bureau of Mines concentrating tests have been performed on two large samples of talus material from the foot of the Klukwan deposit. There is a large alluvial fan of detritus between the cliff and the river which may contain some "hundreds of millions of tons," but samples indicate that the magnetic iron content is extremely low. Two samples (one cu yd each) disclosed iron contents of 11.8 pct and 9.4 pct. It was also learned that the iron-titania ratio is about ten to one throughout the deposit. However, the iron-titania ratio is lowered by grinding and separation. Concentrates of about 63 pct iron and 2.2 pct titania were produced. Concentrate was obtained with almost complete recovery of magnetic iron. Tailing assayed at about 7 or 8 pct iron but practically all of the iron was combined as silicate. It was also determined that at a 20 mesh grind followed by wet magnetic separation, nearly all the tailing could be discarded. However, in order to produce a highgrade concentrate it was necessary to regrind the 20 mesh concentrate to 65 mesh and re-treat it by wet magnetic concentration.

The Bureau of Mines is toying with the idea that the titania may be made into an asset instead of a liability. The Japanese interests continue to perform drilling operations on the property, although admittedly worried about the high titania content.

The second largest Alaskan deposit is on tidewater near Snettisham, an abandoned gold mining camp about 35 miles southeast of Juneau. The Territorial Department of Mines made a magnetic survey of the area, locating four promising zones. Further exploration has been recommended and field work is reported in progress.

If and when the deposits are found to be suitable for exploitation, two markets seem obvious. Japan is a strong possibility, with that nation seeking a stable source of iron ore. The West coast is another possibility. Development of an Alaskan steel industry may not be far fetched. Problems of labor, power supply, and transportation would have to be solved. Hydro power sites can be developed in southeastern Alaska and coal is available for steam power plants. Mechanization might be the answer to labor shortages.

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ANACONDA Copper Mining Co. expects to complete its aluminum plant on the Flathead River, three miles north of Columbia Falls, Mont., by mid-1954. The site has already been cleared and detailed designs of the plant are well advanced. The plant is scheduled to produce some 50,000 tons of aluminum annually. Each of two potlines will be housed in two of the four long parallel buildings. Large Soderberg anodes of the vertical pin type will be used. The property lies along the main line of the Great Northern Railway and is about one mile from Bonneville Power Administration's transmission line from Hungry Horse Dam to Hot Springs and Spokane. Electric power will be brought to the reduction works at 230 kv.

The location of the potlines places the direct circuit terminals at the rectifier station. A fifth building on the easterly side will house the casting shop.

Reynolds Metals Co.'s Bayer plant at Corpus Christi, Texas will supply alumina for the Anaconda plant. After unloading from rail cars at the north end of the plant, alumina will be stored in silos. The flux and reagent storage will also be located at that end, along with the paste plant, boiler house and garage.

Shops, storehouse, office, laboratory, change house, and parking lot will be on the westerly side of the plant. Roofed-over area will cover more than 600,000 sq ft, with some 450 men employed in the plant's operation.

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INDUSTRY will continue to be hampered by a shortage of engineers during 1953, according to Engineering Manpower Commission of Engineers Joint Council The commission asserted that 40,000 new engineering graduates are needed by U. S. industry, but only 23,000 will graduate this year, 19,000 in 1954, 22,000 in 1955, and 29,000 in 1956. The Commission reports, however, that companies have learned to use the engineer's talents more efficiently. Engineers have been relieved of work that can be done by nonprofessionals or technicians.

THE iron and steel industry spent $1,170,000,000 in 1952 for improvements and expansion, with an accompanying rise of 9 million net tons in capacity to 117.5 million tons of ingots and steel for castings. More than $4,600,000,000 has been invested in the seven years following World War II. An average of $1260 has been laid out every minute during those seven years for an increase in capacity of 25.6 million tons.

About $1 billion will be spent during 1953 for an additional increase in capacity of approximately 4 million tons and other facilities such as coke ovens, blast furnaces, soaking pits and rolling mills. Company reports on 1952, starting to appear in an ever increasing stream, indicate that earnings were generally affected by the steel strike. But with some hope for labor peace in the near future, steel mills should continue to make steady gains.

Government reports indicate significant gains in at least eight steel items considered critical. Estimates were based on maximum annual production, but Government officials indicated that industry could not produce all these products at maximum rates.

The eight products mentioned in the report are: Heavy standard rails, hot rolled bars, including tube rounds, cold finished bars, heavy structural shapes and piling, strip mill plates, sheared plates, universal plates, and seamless pipe and tubing.

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U. S. mine production of copper declined approximately 3830 tons in 1952 compared with the previous year primarily due to declining quality of ore produced at some important mines. Arizona, the leading copper state, produced more copper ore than ever before but copper output dropped 5 pct. Arizona produced 44,525,000 tons of ore compared with 42,791,070 tons in 1951. The Morenci mine continued to lead the way in production, although output was 13 pct lower than in 1951. Open pit operations produced more ore than in any other year.

Copper production in Utah and New Mexico, second and third largest producers in the U. S., rose 5 pct, while Montana, next in line, rose 10 pct. Montana's gain was due to beginning of production at Anaconda Copper Mining Co.'s Greater Butte Project during the first half of 1952.

Total copper production for the nation reached 924,500 tons compared with 928,330 tons in 1951. The industry enjoyed a year almost devoid of labor troubles. Copper producers awaited action to decontrol copper. Acting Defense Mobilizer Arthur Flemming announced that Controlled Material Plan allocations will be "open ended," permitting producers to sell copper to any customers who want it, providing all customers with CMP allotments are taken care of first, before those without CMP authorization are served. But copper men awaited the ending of curbs. Many said the laws of supply and demand could not go into effect until controls were removed. The new administration has moved quickly to end what many considered an unfair situation.

CMP will remain intact through the second quarter of this year to facilitate an orderly withdrawal of the law. Customers can start to make arrangements for extra metal, although delivery cannot start before April at the earliest.

Domestic fabricators have been forced to buy 40 pct of their copper needs from abroad. They have paid much more than the 24½¢ asked for copper in the U. S., but have been allowed to pass along only 80 pct of the extra cost.

But with CMP controls on copper running until June 30, and the still tight copper market, no immediate change in the copper situation can be expected. A scramble for any loose copper can be expected, but major producers are said to be considering a voluntary allocation plan to stabilize the market until conditions become clearer. With price decontrol, the copper market fluctuated as producers attempted to determine price.

Copper supplies are expected to be slightly better in 1953, but military and Atomic Energy Commission requirements will be up, according to the Eighth Quarterly Report of the Director of Defense Mobilization, issued January 1. Thus, it is expected that copper availability for consumer products will not show any real improvement this year. Major increases of copper from expansion programs will not be felt until 1954.

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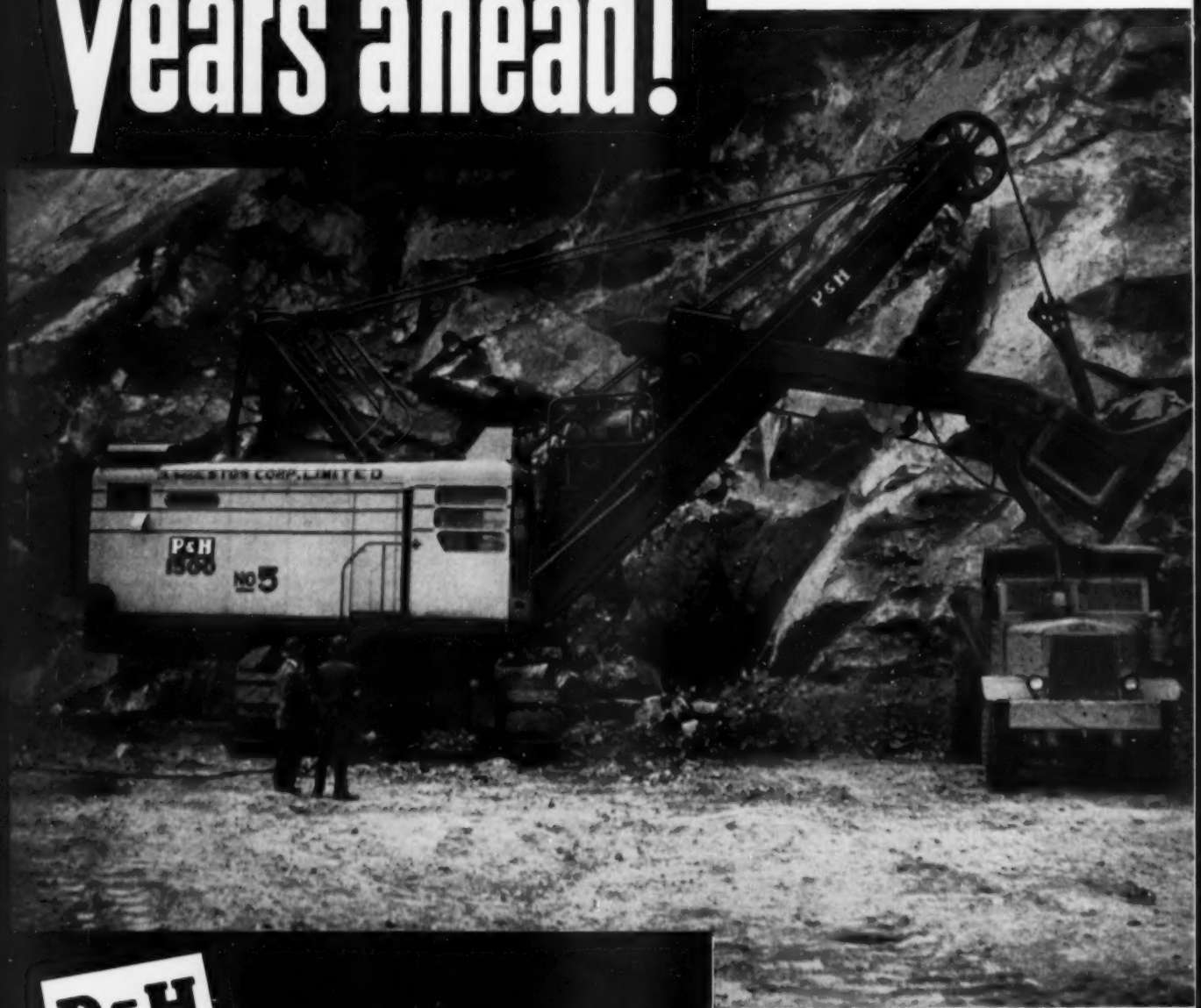
TWO thirds of the $3.5 billion spent for research in the U. S. during the past year came from private companies. The Government spent another fourth, while the remainder came from colleges and universities. The Dept. of Labor, in its survey, *Industrial Research and Development,* received information from some 2000 companies, employing nearly 94,000 research engineers and scientists as of January 1952. Research staffs, including both professional and supporting personnel, totaled 234,000. Average ratio of supporting personnel to research engineers and scientists was about 5 to 1. Average cost per research engineer or scientist was $22,100 in 1951. All research employees, including supporting personnel as well as engineers and scientists, cost an average of $8900. One of the important items reported was that one fourth of all research engineers and scientists included in the study were in classifications most liable for military service. An item of concern is the rapid turnover of research personnel. The average number of research people leaving reporting companies during 1951 was 16.4 per 100 employed at the end of the year. Hardest hit was the aircraft industry, where competition for engineer personnel since outbreak of Korean hostilities has been most severe.

years ahead!



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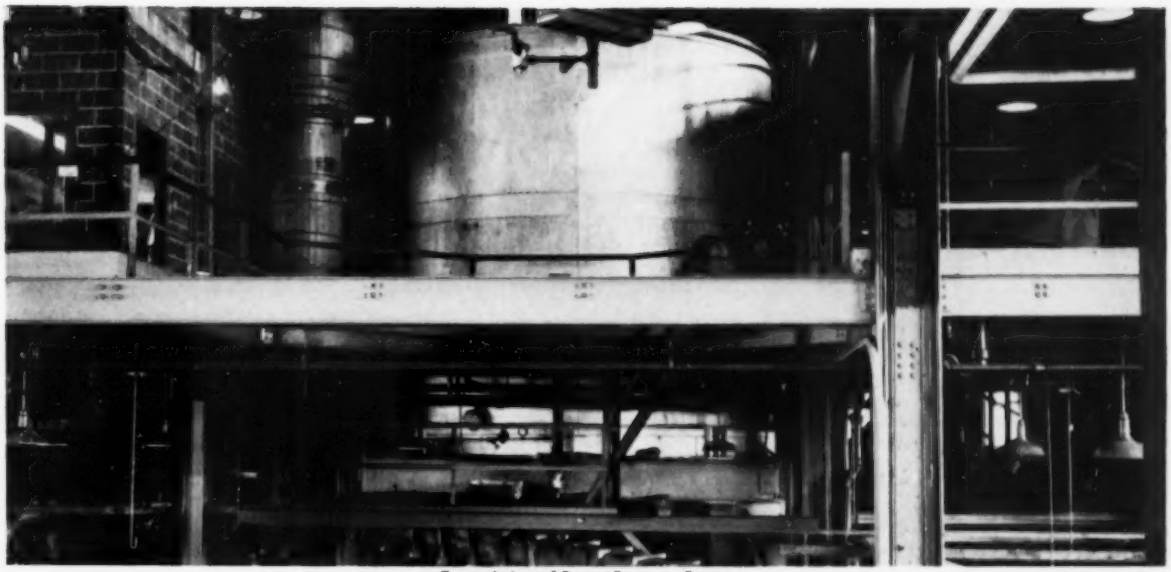
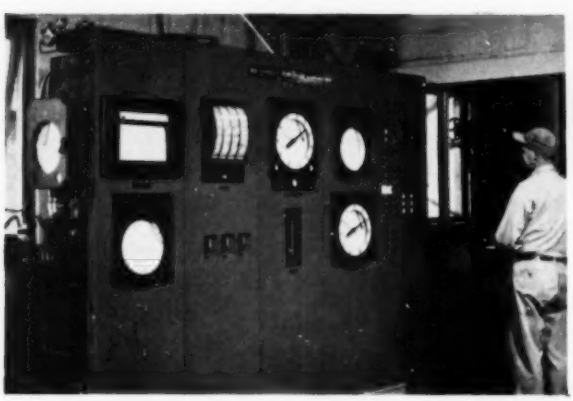


General view of Brown Company Reactor

First Dorrco FluoSolids* Reactor For Producing SO_2 Goes "On Stream" at Brown Company

BERLIN, N. H. In anticipation of a continuing sulfur shortage, The Brown Company at Berlin, N. H. recently installed a Dorrco FluoSolids System to produce SO_2 gas from pyrrhotite. The System at the present time is supplying the entire sulfur requirements of this sulfite pulp mill . . . 11,000 tons of sulfur equivalent per year. A producer of quality pulp for high-grade photographic papers, Brown requires clean, high-strength SO_2 gas to make their sodium sulfite cooking liquor.



Centralized control panel

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The pyrrhotite, recovered from tailings of a nearby copper mine, is received in a moist condition, repulped with water to 70-75% solids, and pumped into the Reactor. By operating close to theoretical oxygen requirements, the strongest possible SO_2 gas is produced, with a black magnetite calcine which is quenched and handled in slurry form. Roast is accomplished in a 16′ inside diameter Reactor with the temperature automatically held at 1650°F.

Clamshell discharging pyrrhotite into blunger

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Gas strength at the top of the Reactor averages 13% SO_2. The gas then passes through a two-stage cyclone system followed by a cooling-scrubbing tower before going to the acid towers. Average chemical analysis of feed and calcine follows:

	Total Sulfur	Sulfide Sulfur	Total Fe
Pyrrhotite Feed	35.7	35.6	49.1
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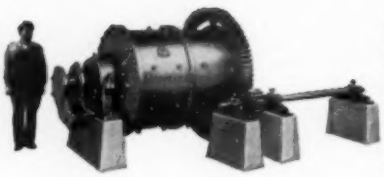
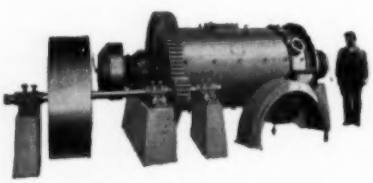
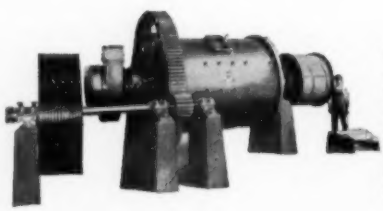


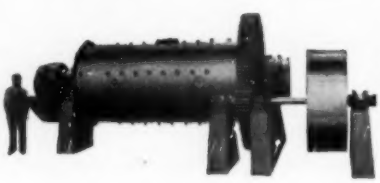
3'x3' Denver Steel-Head Ball Mill arranged for mule back transportation or for adding extra shell sections. Heavy trunnions and bearings allow for adding extra sections.



4'x8' Denver Steel-Head Ball Mill with peripheral discharge for dry grinding. Drum or scoop feed may be used as well as a variety of trunnion type discharges.



6'x8' Denver Steel-Head Ball Mill equipped with trommel screen on discharge. Heads may be welded or bolted to shell sections.



5'x12' Denver Steel-Head Ball-Rod Mill. Heavy trunnion bearings, cast integral with heads, give adequate support to this large mill.

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Drift of Things

WHEN the atomic age broke suddenly upon the world in 1945 with all its terrible implications, predictions concerning the future of atomic energy were as numerous as columnists and often inflated. Today, some seven years since the explosion and probably ten since the Government took over development of nuclear energy, the impact on our way of living has been largely political (except of course for the thousands of people who have become employed in the AEC). Considering the world situation, weapons have come first naturally. But credible progress in the plowshare uses of nuclear energy—medicine, agriculture, power—can also be noted.

The point of this is that Atomic Energy Commission people and our Congressmen, saddled with the awful responsibility of administering and developing the new force, are and have been studying the whole program critically. The big question is whether or not the United States is doing the best it can to harness the energy of the atom. Inextricably tied up in this question is the problem of secrecy for security. America is proud of its industrial know-how which comes part and parcel from the vast reserve of skilled workers, scientists, engineers, and administrators, the product of the system of universal education. It comes from the human desire of the people who have applied their energies over many decades to harvest the resources of our fertile land for homemaking, for comfort, and, when the need has arisen, for self-defense.

Out of fear of the inimical Soviet, protective secrecy has prevented full utilization of this know-how. Many of those intimately connected with the atomic energy program feel that much information, the fruit of billions of dollars spent on research and development, with care can be declassified so that industry can harness nuclear energy for peace. Efforts of the Commission in this direction are bringing the fruits of the Atomic Age closer to reality.

An Advisory Committee on Industrial Information, made up of engineer-editors representing every industry possibly interested, has been functioning for some time. This group has received top security clearance so that it can perform its function of studying atomic energy developments, and processes for possible applicability in industry. The committee has authority only to indicate what information would be useful to industry so that the Commission's security group can consider it for possible declassification and publication in the industrial press.

In the new Congress, the constituency of the various committees is being worked out. The Joint Committee on Atomic Energy is one of these. The rate of progress of the atomic energy program will rest in the hands of this committee. For it political considerations should be waived so that a selection may be made from men of wisdom.

EFFORTS directed at stimulating interest in careers in engineering among high school and college students are taking hold judging from activity in the New York area and the national "Engineers Wanted" campaign sponsored by the Advertising Council. The New York Section AIME has just completed a two-day guidance program for over 175 students from colleges in the vicinity. This program followed the pattern of one held in 1952 by the St. Louis Section. Included in the program was a panel of industry representatives for each branch of the mineral industries which described types of work and career possibilities. Field trips to mines, a copper smelter, and the Fairless works of the U. S. Steel Co. followed discussion. As at St. Louis, all expenses of the students were paid by the Section. This type of interest manifested by future employers in young men goes a long way toward attracting students to the fields of engineering for which such programs are arranged.

Another indication of industry interest in recruiting engineers was the splendid support received from companies for the new edition of the Mudd Fund's "Careers in the Mineral Industries" being published this month by AIME. This support has made it possible to make the booklet more attractive and to give it wider dissemination than would have been possible using only the contribution of the Mudd Fund which has customarily financed this publication.

An exceptional effort is being made in the "Engineers Wanted" campaign of the Advertising Council. The campaign is well planned and it is clearly described in the illustrated booklet "How Your Company Can Help Promote Engineering as a Career." By directing the booklet to the corporations to carry on the campaign, a sound approach is taken because they have the most at stake.

There is no lost motion because each corporation can push the type of engineering which is most important to it. It is as basic as assuming a role in community activities. To mention just a few of the things to be done which are described in the booklet: Distribute pamphlets on vocational guidance and motion pictures on industry stories or job opportunities to schools and the PTA; cooperate with schools in developing posters and other promotional material; present awards and medals in engineering and science; provide speakers for guidance panels for schools, PTA, and other clubs; send representatives for job interviews.

The program is so workable that many corporations are certain to participate. This places an obligation and even some pressure on the mineral industries to get busy. Other professions and businesses have syphoned off many of the fine young people graduating from our schools because they are frequently in the limelight. Interest in engineering is mounting not only because it actually does have such fine career possibilities but also because of the effort of the Engineering Manpower Commission and company recruiting. The mineral industries must look out for its own in this overall campaign for engineers.

FOLLOWING is a clipping from the *Mining and Scientific Press*, Nov. 2, 1907, sent in by Mr. Ernest Blessing, Miami, Okla.:

"The current idea that uranium ores derive their value from their content of radium is probably due to the fabulous value ascribed to radium in many publications—a value based, perhaps, upon prices paid for it as a chemical and physical curiosity, since they are not justified by any uses thus far found for it. The principal known deposits of uranium ores in this country are in Colorado, where carnotite and uranite (or pitch-blende) furnish the product. . . . Experiments made with the metal as a steel-hardening material do not show that any particularly valuable qualities are added to the steel that are not given by the use of other elements that are at present cheaper. Uranium in the form of an acetate is used as an indicator in various determinations in organic chemistry, and other salts are used in iridescent glass and pottery glazes."

We are fascinated by the powers "ascribed" to the press of those days.

John V. Beall

Large and Small Lead and Zinc Miners Reach Unanimity On

SLIDING SCALE IMPORT TAX

FOR two days preceding the Colorado Mining Association annual meeting, officers representing all interests of the lead and zinc producers hammered away at remedies for the ills of the industry behind closed doors at the University Club in Denver. For once in the stormy history of the industry, large and small producers alike came to agreement with an automatic sliding scale stabilization import tax.

The proposal set forth in full on the following page, is not new being merely an amplification of the tariff arrangement whereby tariffs are imposed when the price drops below ceiling. Its advocates claim that it is simple, automatic, and flexible. It will give stability to the industry without subsidies and since no government bureaus will be required to administer it, there will be no cost to the taxpayer.

Price fluctuation, devastating to the miner and custom smelter, is also damaging to the metal consumer who must often set his prices for products a year in advance. In competitive business this is hazardous in a fluctuating market.

Buffer stocks, subsidies, and international trade agreements were given short shift as being wasteful, expensive, and lacking in stability which is the prime objective as there always will be marginal mines at any metal price.

The House Small Business Committee will be receiving the proposal shortly and may well give this measure support as being in the national interest. It will have the advantage of industry wide backing which in the past has been lacking resulting in confusion and faulty legislation.

At an all-afternoon session at the Colorado Mining Association meeting the problems confronting the lead and zinc industry were described by a panel of industry representatives. Chairman Snyder brought these into focus dramatically by cross-examination of the panel. Snyder: Mr. Fitch (Cecil Fitch, Jr., vice president, Chief Consolidated Mining Co., Eureka, Utah) you have a water problem, I think pumping against a 2200-ft head. Fitch: Well sir, its 7000 gpm against a 1200-ft head. Snyder: Are you able to stay open with the present metal prices? Fitch: When you're pumping water you have to stay open or get out of business. Snyder: How many people in Eureka? Fitch: 1500. Snyder: Anyone else operating? Fitch: We are the main employer, there is one other small producer. Snyder: If you shut down what would happen? Fitch: Eureka would be a ghost town.

Following are some of Mr. Fitch's own remarks which pinpoint the difficulties of an important segment of the industry.

SHORT range problems of Utah's lead-zinc mining industry for the past year have centered around prices and labor costs.

Erratic Market Situation

The sharp down-swing in the lead-zinc market last spring has resulted in the closing of many Utah mines and the curtailing of operations in most of the others. As in other areas, the permanent and disastrous results could be the flooding, caving and practically complete loss of the shut-down mines. In many cases where operations are continuing, operating losses are being sustained, exploration and development programs curtailed or discontinued while production is generally limited to available higher grade ore. Obviously this situation can only be tolerated for a short period of time. Lead and zinc mining, like other business, can be carried on only under profitable conditions. The small operator will soon be out of business and the large operator will get sick of riding the ferriswheel of prices and invest his capital in more stable industries. As a matter of fact, a brief examination of the annual reports of many nonferrous mining and smelting companies will quickly reveal that they are rapidly diversifying—going into other fields of endeavor.

In a study recently made by the Utah Mining Association, with assumed prices for lead and zinc at 13½¢ and 12½¢ per lb, respectively, it was estimated that the production of the eleven western states would drop by one third during 1953 and would have less than half of the 1951 value.

Effects on Public

Direct employment, wholesale and retail trade, personal professional services, taxes to local, state and federal governments in the eleven western states will all suffer losses along with the mines. Utah's share of this loss of new wealth is estimated $13.8 million for 1953 compared with 1951.

It is historic that with war or war threat emergencies, metals become scarce in the United States. During those periods the government has urged the greatest possible domestic metal production, meanwhile imposing price ceilings on newly produced metals and allocation and end-use controls on metal consumers. At the same time, uncontrolled foreign prices rise above U. S. ceiling prices and we pur-

chase whatever metals we can get at those higher prices. In this last, so called, shortage we found ourselves competing with foreigners for high priced metals which they were buying with ECA money only to have them discover that their stockpile needs were highly overestimated. When this became apparent they then dumped their excess metals on our market in return for "free" dollars. Although final figures are not available, lead imports for 1952 were about 228 pct of the 1951 imports and were about 70,000 tons greater than the 1950 record. Zinc imports for 1952 were something over 50 pct greater than the previous years. In times between such emergencies when there are metal surpluses, government forgets the previous serious scarcity, forgets the all out effort of our domestic mines, and favors throwing open the doors to the foreign producer. The results are well illustrated by the production and value figures which have been mentioned.

Each one of these cycles further discourages domestic production or development for further production. If not corrected, this condition will continue to the point where domestic production is so small that we will be almost completely dependent on foreign mines. We can then expect a hijacking such as we have experienced in raw rubber and in copper. Not only will the prices be higher than necessary to maintain a healthy domestic industry, but more seriously, we can reasonably doubt whether these metals will be available to us in an emergency or in war.

Labor Developments

The labor situation was an unhappy one for underground mines in Utah during the past year. The Wage Stabilization Board, true to its unstabilizing efforts, in the Utah dispute cases not only refused to consider the underground mining industry's normal district relationship wage pattern, but in its decision reversed long established precedents. It is interesting to note that the Steelworkers benefited in these cases as they did in the Steel case. Forcing deep mines to pay on a scale which can be well afforded by open-pit operations has resulted in a labor cost burden which has hurt all of us and has proven too great for some operations to bear. Some Utah companies still do not have a contract in force for this year and are operating on a day to day basis with little apparent progress being made.

In a competitive economy these two problems will always exist to some extent. We hope, however, that violent price fluctuations can be eliminated by a sliding scale Stabilization Import Tax. It is certain that many small operators cannot stand another ride like this last one. As far as labor relations are concerned we can all have hope that the pendulum will shortly start to swing the other way.

The long range problems of the mining industry relate to the stimulation of exploration and development in new properties and the expansion of present producing mines and mining areas.

Taxation Problems

We do not ask for subsidies. We do, however, desperately need revision of our whole Federal tax structure to remove actual replacement and expansion barriers and to furnish a proper incentive climate for the venture capital so necessary to mineral exploration.

The oil industry with an adequate depletion allowance and with provision for full expensing of exploration costs has, even in our high cost, low yield, Great Basin "oil plays", demonstrated the ready availability of money for "risk ventures" under tax policies designed to give the investor an even break on realizing return of his venture capital.

There's unanimity in the industry on this:

Sliding Scale Stabilization Import Tax on Lead and Zinc

National Lead and Zinc Committee Policy

Adopted in

Denver, Colorado, February 11, 1953

The National Emergency Committee on Lead and Zinc in conference at Denver, Colo., on February 10 and 11, 1953, has concluded that, in the interest of National defense and security, constructive legislation as stated below is needed without delay for the preservation of the Mining Industry engaged in the production of Lead and Zinc.

1. Whenever the respective market prices of these metals are below the prices required to perpetuate the domestic mining industry, a sliding scale stabilization import tax on lead and zinc must be provided.

2. The base prices of lead and zinc on which the tax should be determined should be prices that will yield domestic market prices of 15½¢ per lb, New York, for lead and 14½¢ per lb, East St. Louis, for zinc.

3. These base prices should be adjusted periodically with the BLS index of primary market prices for all commodities other than farm and food.

4. No tax should be assessed when the U. S. domestic market price is equal to or greater than the adjusted base price.

5. In the case of zinc metal and refined lead, the import tax should be at the rate of 1¢ per lb for each 1¢ the market price is below the adjusted base price.

6. The same procedure should apply for zinc and lead in ores and concentrates except that the import tax should be 7/10¢ per lb for lead content and 6/10¢ for zinc content.

7. The legislation should be made effective for not less than the life of the Defense Act, or any extension thereof.

The Committee urges the support of this program by all segments of the Zinc and Lead Mining Industry and all people who believe that the domestic Mining Industry is worth saving, in the interest of National Security.

Packaged Timber Handling At Copper Queen Branch, Phelps Dodge Corp.

Mine timber is cut and framed at the Junction sawmill, strapped in unit bundles, and stored in the different mine yards. Fork trucks do the loading.

by John H. Jett, and Wm. P. Crawford

MATERIAL and supply handling is a major problem at underground mines, especially to those operating through deep shafts with small compartments. Mechanization is necessary, yet in many cases cannot be utilized to fullest advantage owing to the limitations of existing plant design. In cases of this sort, improvisation is indicated and adds to the headaches and enjoyment which most mine operators get out of their work.

Underground operations at the Copper Queen Branch are principally carried on through three vertical shafts; the Junction shaft, Campbell shaft and Cole shaft. Hoisting depths range from 1400 ft at the Cole through 2700 ft at the Junction to 2966 ft at the Campbell shaft. Men and supplies are moved through each of the shafts. Cage compartments are small requiring all material over 5 ft in length to be handled in an upright position on the cages.

Mine timber is the largest supply item, based on volume, handled through the various shafts and requires the most time. Many of the supplies were handled on mine trucks or in cars but timber, due to the size of the shafts, required individual handling on and off of the cages. The hoisting rate per shift was reduced, requiring additional hoist shifts and additional labor was required to load timber on mine trucks for surface and underground transportation. Material handling studies showed timber to be handled a minimum of five times between the Junction sawmill and the underground timber stations.

Exploded view of cage plates show the chair plate (A) and heel plate (B). Strap hook and two removable chair and heel plates are required for each cage.

Special timber trucks, 10 and 12 ft in length, suspended in the cages were used at the Cole and Campbell shafts. Although they were an improvement over handloading and unloading cages and reduced the time that the shafts were tied up in timber handling they had definite disadvantages: the capacity was small, only four 10x10 stope posts or twenty 2x10-in. lagging 6 ft long; truck maintenance was high and they still had to be loaded and unloaded by hand.

The obvious solution was putting up timber in bundles or unit packages and handling a package in approximately the same time as that required for one piece of timber. Chain and wire rope slings were tried and proved unsuccessful. Experiments in packaging timber, using steel strapping, were begun in 1946 and continued into 1947. Approximately 200 bundles of stope timber and lagging were moved underground at the Denn shaft, which was then active. A small hand operated crane placed the bundles in the cage and timber hoists and slings removed the bundles at the different shaft stations. One of the principal difficulties was loading packages on and off the motor trucks which hauled supplies from the Junction mine yard to the Denn shaft, a distance of nearly one mile. The bundles were unloaded by chain block or a timber hoist and moved to the cage by the small crane. The cage loading time per bundle was approximately 5 min

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and the unloading time was 6 min. This was an improvement in point of time over hand loading the cages and proved that packaged timber was feasible.

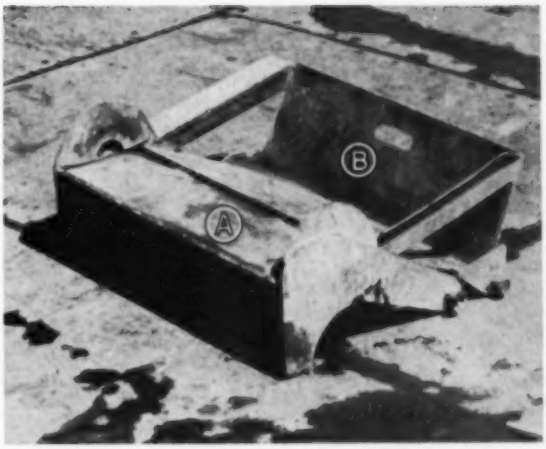
The acquisition of a 7½-ton fork lift truck simplified the motor truck loading and reduced the cost of mine yard handling but did not solve the problem of caging the packages. The experiment was temporarily discontinued and assistance sought from the steel strapping companies. It was a new field for them, the actual bundling was simple but the underground handling presented obstacles. In June 1949, one of the major strapping companies assigned a field engineer to study the problem. A solution was reached after five weeks of field work in conjunction with Copper Queen engineers and underground men and packaged timber handling was inaugurated at the Junction mine in September 1949 and thereafter at Campbell and Cole mines.

Mine timber is cut and framed in the Junction sawmill; strapped in unit bundles and stored in the different mine yards. It is loaded by fork lift trucks to mine trucks and is transferred to the cage by an electric hoist mounted on the cage. The hoist is a hook mounted type, of 1000 lb capacity, with a 20-ft load chain and 110-v single-phase motor operated by tiller ropes. Electrical outlets are installed at each shaft station. A strap hook and two removable chair and heel plates are required for each cage. At present, two cages at each of the three shafts are equipped with hoists and accessory equipment. Load chains are guided by auxiliary sheaves to permit loading and unloading on either side of the shafts and to prevent power kick-off if the chain is inclined at a low angle.

Details of this equipment will be seen in accompanying photographs. The method of loading timber on and off the cage, as follows: Chair and heel plate are placed on the cage and a truck load of timber is trammed to the shaft. The strap hook is inserted under one steel strap and the package is raised to an upright position by means of the hoist. The bottom of the package pivots on the heel plate and rests on the chair, unbalancing the package and making it easily lowered off the cage to a truck.

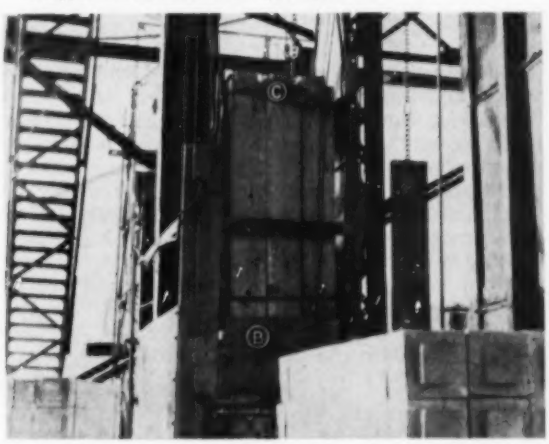
Long timbers are hoisted with a chain sling in place of the strap hook. Timbers up to 16-ft length are strapped and handled as unit packages. Over this

Back view shows bundle caught by heel plate (B). Unbalancing the package makes for easier lowering off the cage to a timber truck. Long timbers are hoisted with a chain sling in place of the strap hook.

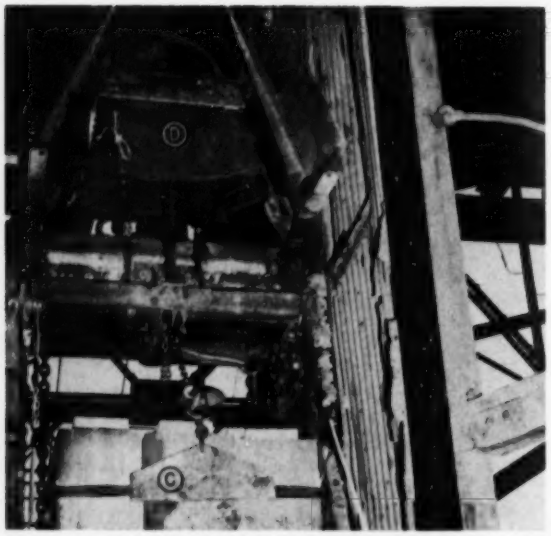


Chair plate (A) and heel plate (B) as they look when assembled on the cage. Bottom of the package pivots on heel plate in loading and rests on chair, unbalancing the package.

A bundle of eight-ft stope posts are hoisted on cage. Loader holds hoist control ropes. Timbers up to 16-ft length are strapped and handled as unit packages.



In this position the bundle is ready to be lowered. Note strap hook (C). Packaged timber is strapped in bundles approximately 30x30 in., and in lengths up to 10 ft.



Details of strap hook (C), electric hoist (D), and hoist chain rigging can be seen. Hoist is a hook mounted type, 1000 lb capacity, with a 20 ft load chain and 110-v single-phase motor operated by tiller ropes.

length they are bored and suspended under the cage. Packaged timber is strapped in bundles approximately 30x30 in. and lengths up to 10 ft. This size bundle fits any of the mine cages and comprises more than 80 pct of the timber lowered. Stope posts, caps and ties are packaged nine to the bundle; 2-in. lagging is packaged 30 to 40 pieces to the bundle. Stringers and long stulls are packaged in single or double layers depending on the timber cross-section. Two straps are used on the majority of the bundles, a single strap is used on small bundles of stulls or stringers.

One and one-fourth in. .035 Heavy Duty strapping was used at first but 1¼ in. .031 Magnus strapping has been in use for more than a year. The Magnus strapping has the same tensile strength as the Heavy Duty and has a larger yield in linear feet per pound. One bundle 30x30 in. requires 22 ft of strapping and two seals. The average overall cost per bundle for strapping material and labor is 55¢. Bundles are strapped in the sawmill at the rate of 12 to 15 per man hr. They are handled on the cages at the rate of 12 to 18 bundles per hr. Formerly, with loose or unbundled timber, the amount handled per hr was equivalent to 2 to 3 bundles.

Monthly timber consumption for all underground mines in Bisbee ranges from 350,000 to 600,000 board feet. Since the inception of packaged timber handling in 1949 the amount of labor assigned to mine timber handling both surface and underground has been reduced. Packaged timber is easier to transport underground and has reduced tramming hazards. Approximately 20,000 bundles have been handled without a single strap breakage or seal pull-out. At present, four fork lift trucks are in use at the Copper Queen Branch; one each in the Campbell and Cole yards and two in the Junction yard. Rail shipments of timber are unloaded and

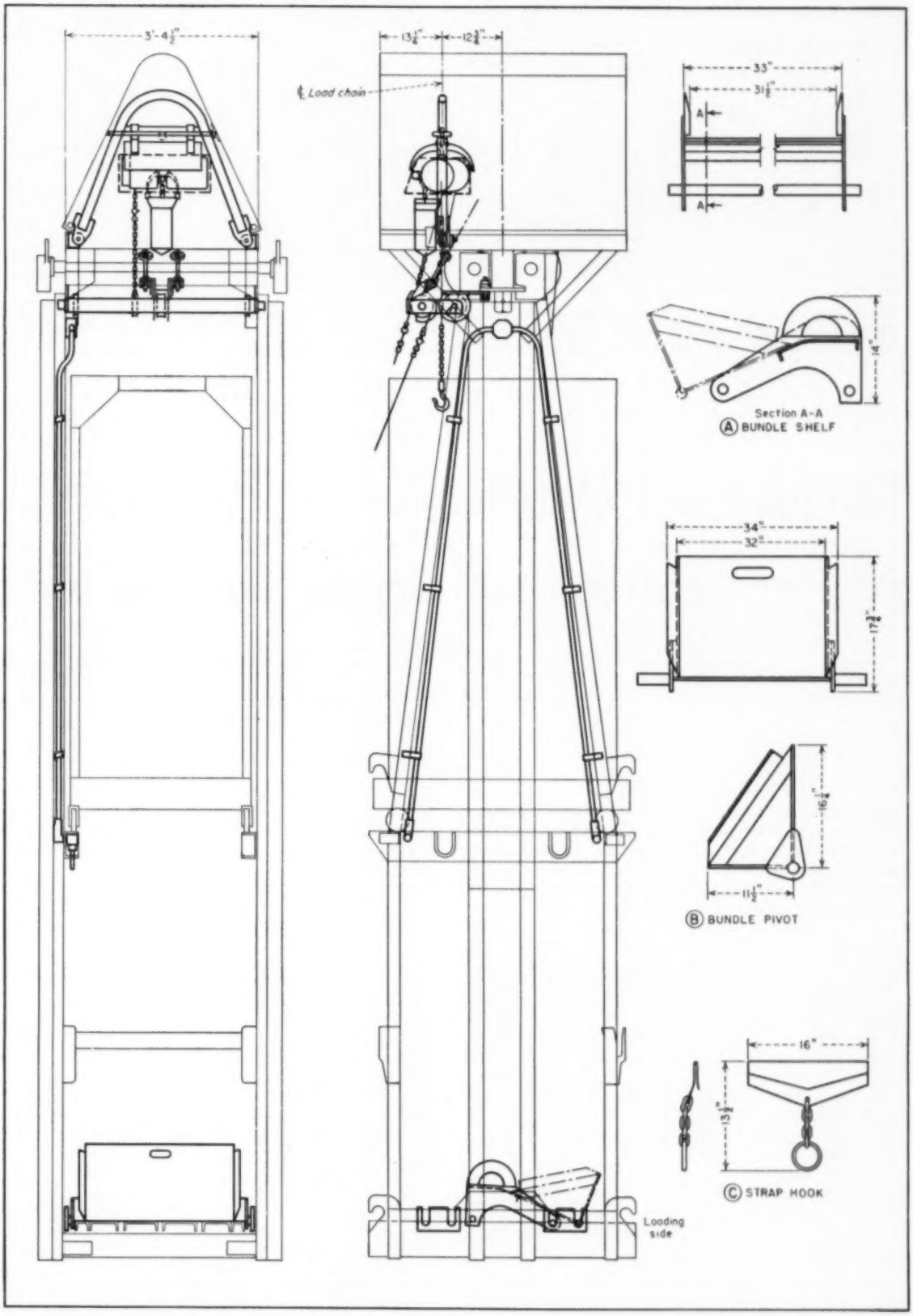
Man underground loads timber bundle on timber truck. Note receptacle (E). Timber is easily unloaded to timber truck because of unbalanced loading on cage. Trailer power cable to hoist remains on cage with a receptacle on each underground station.

piled by fork truck or motor crane. Timber is moved to the Junction sawmill by fork truck and passed through the mill by means of a timber transfer, which was designed and built in the Copper Queen shops, and live rolls.

Pipe, ladders and small rails are bundled in unit packages and handled in a similar manner to timber. Strapping has reduced damage to pipe threads. Other materials handled underground in unit packages are cement, lime and sacked gunite. Incoming shipments of cement and lime are loaded on pallets in the railroad car and move thereafter as unit packages. Gunite is mixed in a central mill and sacked in ½-cu ft packages and stored on pallets. Labor of handling and sack breakage for all these materials has been materially reduced.

OPPOSITE PAGE shows general arrangement of cage equipment required to send packaged timber underground. Chair plate (A) and heel plate (B) provide pivot for timber, package is unbalanced to ease unloading. Strap hook (C) is utilized for packages, long timbers are also handled by hoist, but chain sling is used. Photographs above show hoist (D) and receptacles (E). Power for 110-v hoist is provided by cable plugged into receptacles at each underground station.

CAGE WITH ACCESSORY EQUIPMENT FOR TIMBER HANDLING



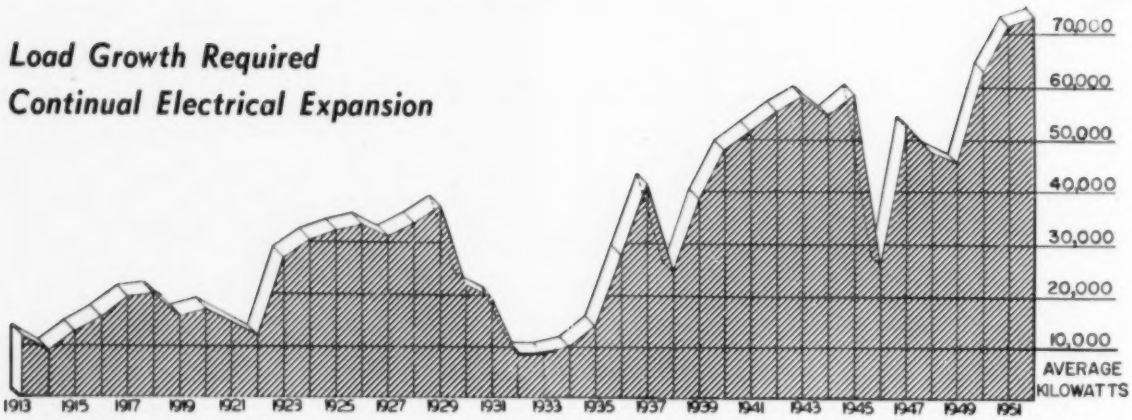


Fig. 1—Progressive growth of the company electrical system is shown in the graph. All power for milling is now generated in a 100,000-kw steam plant completed in 1948. Original power distribution had been at 440 v. Revamping and expansion of this system to 13,800 v, with step-down to 480 v, delayed first by disturbed economic conditions, then by World War II, was undertaken in 1950.

Electrical and Metallurgical Improvements At Kennecott's Utah Copper Division Mills

by R. J. Corfield and A. G. Johnson

MODERNIZATION of the entire electrical system and improvement of flotation process efficiency is the twofold goal of the improvement program underway at the Arthur and Magna concentrators of the Utah Copper Div., Kennecott Copper Corp. Copper ore from the Bingham pit goes to these two concentrators where an interlocking program calls for complete electrical modernization and increased flotation process efficiency through additional capacity and flow scheme revisions. Flotation plants at the two mills are essentially duplicates, and while this article deals with the changes at the Arthur mill similiar improvements are being made at the Magna mill.

Prior to 1913, all power for the Magna plant was generated in the Utah Copper steam power plant at Magna, which had a capacity of 8400 kw. Increased loading during 1911 and 1912 exceeded the capacity of the power plant, making it necessary to either enlarge the plant or obtain power from another source. The economical choice at that time favored an outside source, therefore in 1913, a power contract was entered into with the Utah Power and Light Co. Progressive growth of the Company system is indicated in Fig. 1. The sharp upward trend, beginning in 1935, compelled a certain amount of increased electrical capacity. Subsequent modernization involved addition of 6000 kva transformer capacity, installation of metal clad air-break switchgear to eliminate all 600-v oil circuit breakers, and rerouting part of the 440-v feeder system.

In 1913 power was distributed at 440 v, although serious consideration had been given to 600 and 2300-v systems. Higher voltages were eliminated because of the presence of considerable moisture resulting from the wet processing of ore over tables. The amount of power to be distributed at that time was within the economic range of a 440-v system. However, as indicated in Fig. 1, load growth required continual expansion as rapidly as possible. This was accomplished by adding transformer and switch capacity together with an extensive 440-v cable distribution system. By 1937 this low voltage system had become inadequate and complete revamping was seriously considered but was set aside because of economic conditions. Again in 1941 the program remained dormant due to World War II.

As the trend toward additional power continued and it became more and more evident, through system failures, that present electrical facilities were

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inadequate, a decision was made, in 1950, to completely abandon the original system and replace it with a new, up-to-date distribution system. Additional power requirements of the contemplated expanded flotation plant were also a factor.

Power Distribution

The new distribution system at each mill will, upon completion, consist of one primary substation having a normal capacity of 40,000 kva, receiving power at 44,000 v and transforming it to 13,800 v. This power will be distributed to fifteen 2500 kva unit substations where it will be further transformed from 13,800 to 480 v. Unit substations with associated air-break switchgear are centrally located on each of the 13 mill sections and will service fine crushing, ball mill grinding and flotation departments. The other two unit substations service the coarse crushing department and maintenance shops.

The main 480-v secondary feeders are 600,000 cir mils, 3-conductor, rubber insulated, neoprene jacketed cables installed in conduit or open racks.

All power for milling operations is generated in a modern steam plant, completed in 1948, having an installed capacity of 100,000 kw. The flotation process consumes about 15 pct of the total power used by the Arthur plant. This power is presently being supplied from a substation receiving power at 44,000 v which is transformed to 440 v for use at the flotation machines. Present flotation machines are directly driven by 10 hp, 600 rpm, standard vertical induction motors. There are 595 such motors in service at the Arthur concentrator.

Flotation Motors

The agitating mechanisms of 720 new and larger flotation machines, now being installed, are V-belt driven by individual 15 hp, 1800 rpm special high efficiency vertical induction motors. The power input per machine is roughly twice that of the old machines. Facilities to supply this additional power are incorporated into the new power system.

A typical flotation feeder circuit is shown in Fig. 2. Each row of flotation machines requires 16 motors. Each motor is controlled by a combination line starter. A 3-conductor No. 10 rubber insulated, neoprene jacketed cable serves as a motor feeder. These feeders terminate in a water tight receptacle with the plug being a part of the motor connection. Such a device expedites the changing of motors or agitating mechanisms.

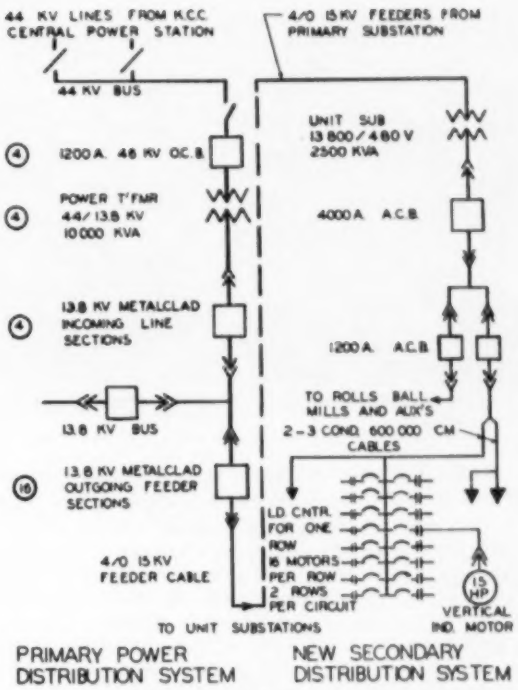
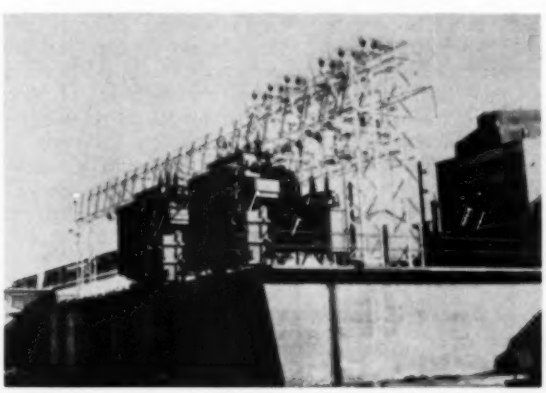

44 KV LINES FROM K.C.C. CENTRAL POWER STATION
4/0 15 KV FEEDERS FROM PRIMARY SUBSTATION
44 KV BUS
UNIT SUB 13 800/480 V 2500 KVA
4
1200 A. 46 KV O.C.B.
4
POWER T'FMR 44/13.8 KV 10 000 KVA
4000 A. A.C.B.
4
13.8 KV METALCLAD INCOMING LINE SECTIONS
1200 A. A.C.B.
TO ROLLS BALL MILLS AND AUX'S
2-3 COND. 600 000 CM CABLES
13.8 KV BUS
16
13.8 KV METALCLAD OUTGOING FEEDER SECTIONS
LD CNTR. FOR ONE ROW 16 MOTORS PER ROW 2 ROWS PER CIRCUIT
4/0 15 KV FEEDER CABLE
15 HP
VERTICAL IND. MOTOR
TO UNIT SUBSTATIONS
PRIMARY POWER DISTRIBUTION SYSTEM
NEW SECONDARY DISTRIBUTION SYSTEM

Fig. 2—Electrical feeder circuit for typical flotation section shows progressive step-down with new system.

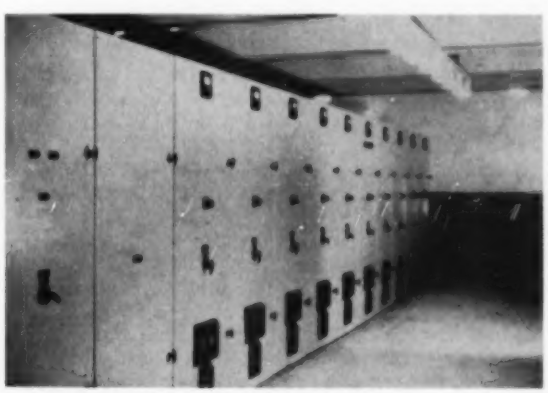
Starting and stopping of flotation motors is accomplished by a centrally located control console having 128 pushbuttons arranged in 8 rows of 16 buttons. Stop buttons are arranged to be locked for the protection of maintenance personnel.

It can be said that this new electrical distribution system, ultra modern in every detail, embodies not only electrical advantages over the old system, but provides the features of flexibility and sectional isolation which permits constancy of operations so necessary to maintained metallurgical efficiency.

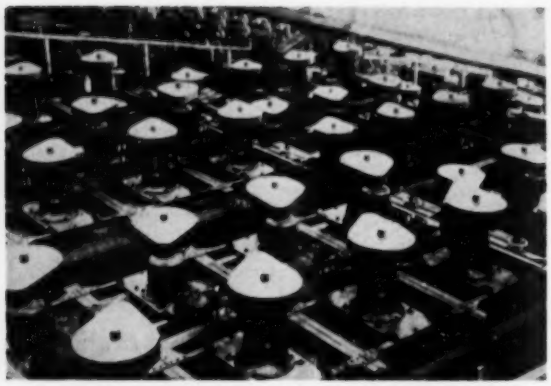
One of the contributing factors in influencing the decision to modernize the electrical distribution system was the additional power requirements of the contemplated expanded flotation plant. The new flotation system, now under construction, comprising new and larger flotation machines as well as flow revisions, was the result of several years of research



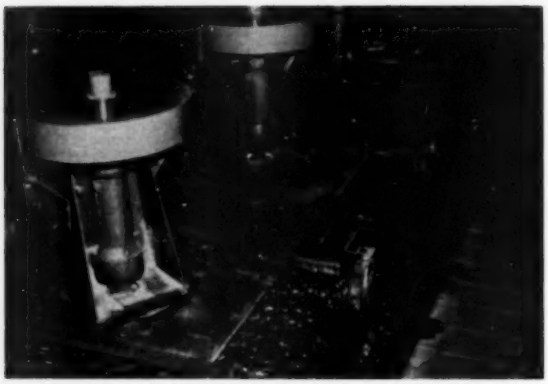
Exterior view of 44 to 13.8 kv primary substation showing main transformers and primary switchgear.



Interior view of primary substation showing part of the 13.8 kv switchgear which feeds 15 substations.



One section of the new 62-in. trough type Fagergren flotation machines.



Closeup of three of the new, more compact, Fagergren flotation machines.

—research which had as its aim the increased recovery of minerals from heading ore.

Metallurgical Research

With the change from gravity to flotation concentration, much was gained in both mineral recovery and grade of concentrate produced. Further refinements in the treatment process produced a more selective practice whereby a major portion of the iron pyrite could be eliminated from the copper concentrate.

In the middle nineteen thirties investigations were renewed to develop a suitable means of recovering molybdenite (MoS_2) known to be present in Utah copper ore in small amounts. The change to sodium dicresyldithiophosphate as a collecting agent in the copper flotation circuit, was mainly responsible for floating a good portion of the molybdenite along with the copper. Subsequent separation of molybdenite from the copper minerals is accomplished in the molybdenite recovery plant. For the most part the recovery of molybdenite in the copper flotation circuit remained under 80 pct from the time of starting molybdenite recovery until the present.

Molybdenum Recovery

During the years from 1937 to date a large amount of laboratory and plant experimental flotation work has been conducted in attempts to improve molybdenite recovery in the copper flotation circuit. Many different collecting agents and combinations of reagents were tried. The results invariably showed that any reagents incorporated into the copper circuit and found effective in floating more molybdenite would harm copper recovery. One such reagent, known as *burner oil*, was especially effective. Attempts to add burner oil to the copper circuit always hurt copper recovery when added in amounts sufficient to be effective in increasing molybdenite recovery. Additions to the last flotation machine or two of a particular row gave like results.

Continued testing pointed to the retreatment of copper plant tailing as the solution to improved molybdenite recovery. This method would not in any way interfere with copper recovery and would allow the use of any reagent specific to molybdenite.

In September 1947, preliminary plant experimental flotation tests were conducted following the scheme of retreatment of copper plant tailing. To this end the tailing of one row was pumped to the head of another row, thus providing a means by which the second row would act as the retreatment row. A series of tests were carried out to determine the most satisfactory reagents to employ and also the number of retreatment flotation machines needed. These tests were conducted utilizing two rows of the 56-inch Fagergren step-down type flotation machines.

Metallurgically, the retreatment process was very attractive as evidenced by additional recoveries made. Not only was there a beneficiation due to improved molybdenite recovery but added quantities of copper, gold and silver were extracted.

Research proved that it would be advantageous to consider a row of flotation machines longer than those currently in operation. Consequently, it would be necessary, at the Arthur concentrator, to build a new flotation plant equal in size to the present building in order to accommodate the contemplated retreatment plant. This led to an investigation of two trough-type flotation machines which would have a greater tonnage capacity and permit in effect a larger number of machines to be installed in a given space. This is accomplished by placing the agitating mechanism in a tank common to both agitation and flotation compartments.

Neither of the flotation machines tested showed any metallurgical advantage over the step-down type 56-inch Fagergren 18 machine row. They did, however, have the advantage of compactness.

The machine selected for the Arthur flotation plant was the trough type, 62-inch Fagergren machine. Each row of flotation is made up of 16 of these machines or cells arranged in four tiers of level tanks, each tank containing four flotation cells. The first eight cells replace the nine roughing cells of the step-down 56-inch Fagergren row. The last eight cells, the first of which is an emulsifier, represent the retreatment or scavenger circuit. At the present time the scavenger rougher concentrate is cleaned and the cleaner concentrate combined with the molybdenite laden tailings produced on section A of the molybdenite recovery plant. Section A treatment involves the use of dextrine in the preliminary separation of copper minerals and molybdenite. The scavenger cleaner tailings are circulated to the head of the retreatment section of each row. One row of the new machines will handle 1000 to 1100 tons per day as compared to 750 to 800 tons for the replaced 56-inch Fagergren rows.

The ultimate in metallurgy has not, in all probability, been reached with the conversion over to the new flotation scheme. Research will be continued in an ever hopeful search for better reagents and more efficient machines and processes.

Diatomaceous Earth———

Diatomite quarry for Eagle Picher's Celatom plant, Clark, Nev.

Non-Metal of a Thousand Uses

C. V. O. Hughes, Jr.

DIATOMACEOUS earth is probably the most widely used, and certainly the least well publicized, of the important non-metallics of the United States. The very multiplicity of the names by which this whitish earthy sediment is described has served to confuse its identity, and to dissipate a realization of its full importance.

Variety of Names

Among old-timers in California, this earth is called "chalk rock", in Oregon and in Washington, it is known simply as "silica". Among mineralogists, and by many of the new-comers to the industry, the term "diatomite" is favored. The railroads, perpetuating an old error, describe it as "infusorial earth" in tariffs. These are by no means all the names whose use has disguised, concealed or confused the diatomaceous origin of the material described. In the United States today, diatomaceous products are marketed under trade names, of which some of the most widely known are Celite, Dicalite, Celatom, Kenite, Chick-Bed, and Airox.

A current example which demonstrates the unrecognized everyday presence of diatomaceous earth is the trade name "De-Moist". De-Moist is sold in large department stores, it is advertised and sold by the largest mail-order house in the world. But very few people realize that De-Moist is a calcined granular diatomaceous earth product, impregnated with a deliquescent salt. Diatomaceous earth acts both as the high-surface-area support for the salt, and as the high-capacity blotter that holds the absorbed water safe against dripping.

Common Uses

Perhaps the reader brushes his teeth with a tooth powder containing diatomaceous earth as a mild non-scratching abrasive. The stove in his home and the safe in his office may be insulated with diatomaceous earth. The heads of the matches he strikes are guarded against afterglow by a fine diatomaceous filler. The quality of the paper that he reads, and on which he writes, is improved by a diatomite paper filler. The polish that cleans his silverware, and the one that brightens his car, are sure to owe their effectiveness to diatomaceous earth.

C. V. O. HUGHES, JR. is Asst. Manager, Virginia-Carolina Chemical Corp., Nichols, Fla.

Filteraids

White sugar, familiar household article, depends on diatomaceous earth filteraids for its end purity. Diatomaceous filteraids touch our everyday lives at almost every turn. Many common fruit juices, and practically all beers, wines and whiskeys are filtered with powdered diatomite. Penicillin, streptomycin, aureomycin, chloreomycetin and other anti-biotics require thousands of tons of diatomaceous earth each year to bring them to their final purity.

Synthetic fibers are formed from solutions clarified by diatomaceous filteraids. Without this clarification, minute foreign particles present in the fibers would weaken them locally, and cause them to break during the critical operation of spinning. And, whether or not our clothes are of synthetic fibers, when they are dirty, they go to a dry cleaner who puts diatomaceous earth in his used dry-cleaning fluid, and then filters it to make it fit for re-use.

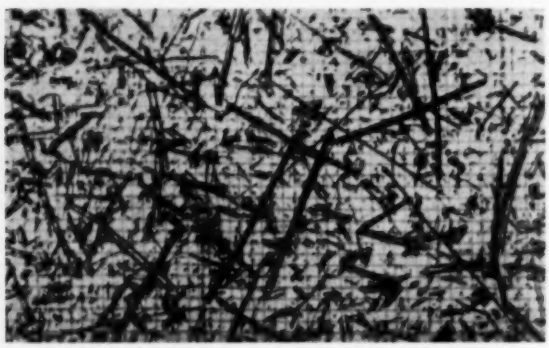
Filteraids are now the most important type of product made by the industry. Over half of the tonnage of diatomite processed goes into filteraids. Because of the high tonnage, and also because specialized products such as filteraids bring premium prices, by far the greater part of the monetary value of diatomaceous products is in filteraids. But filtration is by no means the only important industrial use of diatomaceous earth.

Powdered diatomite is widely used in paints and varnishes as a flatting agent and extender, and to

provide *tooth* in primer coats. Concrete and stucco mixes often include diatomaceous earth to give improved *workability*.

In steel mills and other important high-temperature operations, diatomaceous earth bricks and aggregates provide most effective insulation for exacting applications. In agriculture, diatomite powders are the coating agent that keeps individual particles of chemical fertilizers from sticking together. These same powders are a major ingredient of insecticide mixtures, acting as the inert carrying agent that causes the dust to disperse effectively.

In the liquid air, and liquefied gases industries, diatomaceous earth is used as the absorbent inside the steel cylinders. In war, diatomite finds use as a filteraid which makes safe drinking water out of impure and impotable local water. Diatomaceous insulating aggregates also travel on naval vessels as an essential part of the boiler foundations.



Micrograph of high grade filteraid earth from stratum at Palos Verdes, Calif., which contains only needle-like diatoms.

Filteraid, filler, admixture, absorbent, insulator, coating agent and conditioner, mild abrasive and polish, carrier and support—diatomaceous earth is very nearly all things to all people. It is perhaps small wonder that, with its uses spread so far and so thin, there has been a general lack of recognition of the overall importance of this mineral, and of the industry based upon it. The emphasis on use of the various trade names has fostered this lack by directing attention away from diatomite in general, and toward some specific brand.

Production

The United States is the world's leading producer of diatomaceous earth. It supplies its own needs, and exports filteraids to the British Isles, continental Europe, South America, Cuba, Hawaii, and Australia. Total production figures are never released on an annual basis; the Bureau of Mines is permitted only to publish three-year averages. The industry is growing constantly and rapidly, so that any production figures become badly outdated in a few years. A recent estimate is that output of finished materials has reached the rate of 300,000 short tons per year. The writer believes that this does not take into account all producers, and that the U. S. production rate is fully 350,000 tons.

Localities and Values

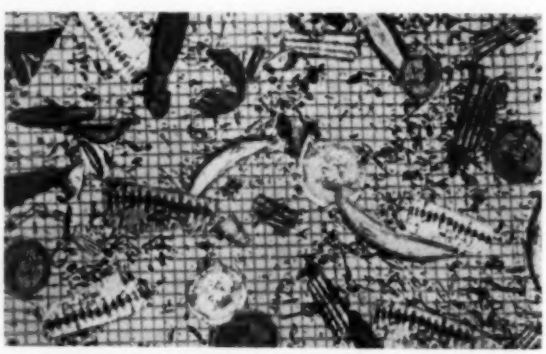
Virtually all of this production is from the Western states. California accounts for about 80 pct of U. S. tonnage. Nevada is the next largest producer; Oregon is probably still in third place; Washington, with several smaller operations, is close behind. Other Western states that have in the past produced diatomite are Idaho, Utah, and Arizona.

Diatomaceous earth occurs in all of the 48 states, although most of them have only small and impure deposits. In the Eastern states, large deposits of low-grade earth are in Maryland and Virginia. Florida has generally small but higher-grade deposits. Maryland was the largest producer of diatomaceous earth until California took over this position shortly after 1910. However, in 1949, the only four states reporting production of diatomite for sale were California, Nevada, Oregon, and Washington.

There is no open market in diatomite; it is sold by contract, or from warehouse stocks maintained by the manufacturing companies. Very little of the tonnage of diatomaceous products is sold for less than $30 per ton, carload lots, f.o.b. mill. This figure represents only a minimum price for filler grades; filteraids bring a considerably higher price. It can be seen that the industry, operating at a level of about 350,000 tons per year, represents an annual product value of well over $10 million. The correct figure may be about $15 million per year.

Producing Companies

By far the two largest producing companies in this field are Johns-Manville Corp. and Great Lakes Carbon Corp. Between them, these two account for about 85 pct of all U. S. production. Johns-Manville's Celite Division has production facilities wholly concentrated at Lompoc, Calif. Here, in what amounts to several separate plants grouped together, it produces powders, aggregates, and bricks.



Micrograph of high grade filteraid earth from Terrebonne, Ore., shows large elongated and disk type diatoms.

Great Lakes Carbon Corp.'s Dicalite Div. operates four plants: one at Lompoc, Calif., one at Torrance, Calif., one at Terrebonne, Ore., and one at Basalt, Nev. Another Dicalite plant, at Kittitas, Wash., was abandoned in 1948. With the opening of its newest plant at Lompoc in April of 1952, Dicalite may have become the largest producer of powdered diatomaceous products. However, it produces no insulating bricks, and only minor tonnages of granular products and aggregates.

Eagle-Picher has one plant, located at Clark, Nev.; its Celatom products are made there. These products are both granular and powdered, calcined and uncalcined. Most of the powders are for fillers, conditioning agents, and polishes; but some are of filteraid grade.

Other producers are the Quincy Corp. of Quincy, Wash., making both calcined and uncalcined powders; Chick-Bed Company, of Fernley, Nev., making

Diatomaceous earth deposits and quarries at Palos Verdes, Calif. Portion of Dicalite's processing plant shows in lower left. Terrain is typical of California's Coast Range area. Deposits are largely located in area of rolling knolls, amidst farms and dairy area.

granular products with uncalcined powders as a by-product; Kaiser Mining & Mfg. Co. of Yakima, Wash., making uncalcined powders; and Airox Corp., at Casmalia, Calif., making calcined aggregates. Several other minor operations intermittently turn out small tonnages in California, Nevada, and Washington.

Deposits

The various deposits of diatomaceous earth have much in common, and yet points of marked dissimilarity. All are, of course, sediments—sediments formed by the deposition of skeletons of a microscopic aquatic plant, the diatom. The tiny diatom lives today, as it has for millions of years past, in great profusion in all our waters, both salt and freshwater. It has been called "the grass of the sea", because of its fundamental position in supplying food for other sea life. It is a single-celled plant related to the Algae. It differs from most plants in that it forms a skeleton. When the diatom dies, this siliceous skeleton sinks to the bottom of the water in which the diatom lived.

There are some 10,000 recognized varieties of diatoms. The predominant varieties to be found in any one deposit are dependent, among other things, upon water temperature and salinity at the time of diatom growth. The most common types of freshwater diatoms differ greatly from the most common types of saltwater diatoms. On the other hand, some freshwater types closely approximate diatoms of marine deposits.

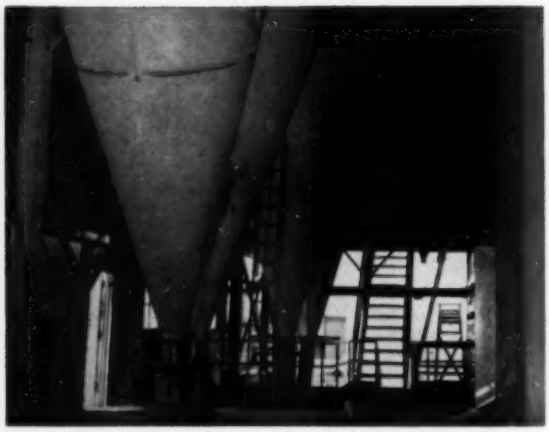
Purity of the deposit is mainly dependent upon the presence or absence of settleable contaminants coexisting in the water with the diatoms. Most common impurities are clay, calcium and magnesium salts, and sands. Other impurities, not always objectionable, are skeletons of microscopic animals living in the water with the diatoms, and organics. The animal skeletons are generally found in marine earths, where fragments of Radiolaria and of sponges are not uncommon. Organic impurities are not necessarily harmful, and may indeed be helpful if the earth is to be calcined. Only the heaviest type of organic residual, such as asphaltic compounds, may be objectionable in earth that is to be calcined.

Diatomaceous earth deposits vary in thickness from a few feet to several thousand feet, and in area from a few acres to many thousands of acres. The earth may be unconsolidated, as in a bog deposit, or so compacted and hardened as to sound like chinaware when struck or shattered. The more usual earth, while perhaps possessing toughness, is soft enough to be crushed in the hand with moderate to strong pressure.

Overburden depths in the workable deposits may be as low as one or two times the depth of recoverable and usable diatomite. Quarries have been worked in overburden as thick as 10 to 15 times the depth of economically usable earth beneath. Deposits are known with still higher overburden ratios. It has been said that these might be mined by underground methods. However, diatomite is a weak, tricky and dangerous material in which to open a stope under any considerable roof pressure.

Diatomite beds are sometimes relatively undisturbed and of rather uniform thickness. But often, as in the California coastal deposits, they are badly folded, with rapid lateral variations in thickness (i.e., pinching and swelling). In most instances, diatomaceous earth is soft enough to yield to pressure by crumpling, folding and pinching. Faulting is not generally serious. Notable exceptions are some of the harder freshwater earths found in the Nevada desert areas. Continuous strata or discontinuous lenses of hard opalized or chertified material are often associated with diatomite, in both marine and freshwater deposits. In some of the Washington and Nevada deposits, this opalized material may assume the form of unpredictable horses upthrust into the diatomaceous earth from the bottom. This material is highly undesirable, not only from the standpoint of the mining problems posed, but from the later processing difficulties encountered in its removal.

Typical diatomaceous earth (if, indeed, any earth may be said to be "typical") may analyze from 80



Cyclone separators for classifying finished product at Eagle Picher's Celatom plant, Clark, Nev.

to 90 pct SiO_2; with ignition losses (combined water plus organics plus CO_2) of 4 to 8 pct; Al_2O_3, 3 to 5 pct; Fe_2O_3, 1 to 2 pct; CaO plus MgO, 1 to 3 pct; alkali oxides, 1 pct; other constituents, negligible. The silica in diatoms is amorphous and contains combined water; it is classified mineralogically as opal. It is generally described as "inert", and so it is, with respect to most chemicals, and particularly acids. The natural earth does react strongly with the alkalis, and also with lime. This reactivity is greatly reduced by calcination, and almost completely inhibited by certain types of flux calcining.

Diatoms vary greatly in size, from variety to variety, and from individual to individual within the same variety. The true size range of diatoms has been somewhat obscured by use of the term "effective particle size". Here "effective" means a fictitious size calculated from Stoke's law, using diatom settling rate in water and true specific gravity to make the calculation. Actual measured sizes of diatoms are less than 10 microns for a very small diatom; 20 microns for a moderately small diatom; 30 to 50 microns for a medium-sized diatom; 100 microns for a large diatom. The writer has seen very large diatoms at least as long as 300 to 400 microns. These occur both in the elongated diatoms of Oregon freshwater earth, and in the needle-like diatoms found in selected strata of California marine earth. In more commonly understandable terms, a very small diatom may measure 0.0001 in. across; a very large one may measure 0.01 in. across.

Diatom shapes and internal structures are greatly varied and intricately beautiful. No attempt will be made here to describe these structures. Only visual reference to photomicrographs serves adequately to demonstrate the variety and beauty that led one researcher to describe the tiny skeletons as "jewels of the sea".

Marine vs. Freshwater Quality

There is no question that the marine earth deposit at Lompoc, Calif., is far and away the largest, the most versatile, and the most consistently good quality deposit in the United States. This statement might be broadened to include the whole world, insofar as is indicated by our present knowledge of the discovered commercial deposits of the world. However, there are other fine deposits besides Lompoc's, of both marine and freshwater origin.

It was at one time fashionable to stress the division of diatomaceous earth deposits into marine and freshwater types, and to question the suitability of any freshwater earth for the production of filteraids. This viewpoint did have some basis in that many freshwater earths are formed from the skeletons of small, heavy-walled, barrel-shaped Melosira diatoms. However, where the freshwater diatom is of suitable size, shape, and wall thickness, it will make a filteraid fully comparable to that made from any marine earth.

The Terrebonne, Ore. operation of Dicalite Div. of Great Lakes Carbon Corp. is based on a freshwater lake deposit. It was the first large-scale diatomaceous earth operation to prove that good filteraids of certain types could consistently be made from freshwater diatoms. Its performance did much to lay to rest the old claim that only marine earths were suited to filteraid production.

Another extremist view held that almost any deposit of diatomite could be made to yield good filteraids, if only particle size distribution of the products were held within certain ranges. There is considerably more to the uniquely satisfactory performance of diatomaceous earth then merely particle size distribution. This in no way detracts from the recognized importance of achieving sharp particle size classification even with a suitable earth.

Processing

There is not space in this paper to treat the processing of diatomaceous earth in any detail. Furthermore, there are many trade secrets, or supposed secrets, involved in the processes used. However, there are many general principles and practices in diatomite production techniques that are matters of public knowledge.

Diatomaceous earth as it comes from the ground is usually wet and chunky. Because of its characteristic high porosity, the earth holds high percentages of water. In some freshwater earths, occurring in desert areas and exposed to sun and wind, surface samples have shown only 10 to 15 pct moisture content. However, on fresh surfaces at working depths, this same earth may show water contents of 25 to 35 pct. Marine earths, even where stockpiled in the dry California summer, rarely average as low as 25 pct moisture. Fresh from the ground, they may run 45 to 50 pct moisture; in wet winter weather, over 50 pct. Poorly consolidated Oregon freshwater earth, protected by a soil cover, has been known to run 55 to 65 pct moisture.

It may be seen from these figures that one major consideration is drying. With earth at 50 pct moisture, it is necessary to evaporate one ton of water for every dry ton of earth recoverable. This is generally accomplished in a flash-drying, air-conveying and air-classifying system. The light weight of the earth, and the ease with which it is milled, make it peculiarly adaptable to air-conveying.

The earth is classified in the same system in which it is dried, milled and conveyed. This classification goes at least as far as the removal of coarser or heavier impurities, collectively described as *grit*. This includes sand, rock fragments, fine pumice, volcanic ash, and possibly roots and sticks (if the bed being milled is near the surface). Whether classification at this stage needs to go further than grit removal is a question not fully agreed on between operating men in the industry.

After drying, milling, and some classification, the powdered diatomite may be considered a finished product, or it may be an intermediate product ready

for the further process of calcination. If it is to be packaged as a finished product, it has definitely been classified beyond the point of grit removal alone, and it is termed a *natural* powder.

There are two broad classes of powdered calcined diatomaceous earth. One is the straight-calcined, or *pink;* the other is the flux-calcined, or *white*. The straight-calcined product has been calcined without addition of any reagents. The trace of iron in the earth generally gives these products a pink color.

To make flux-calcined products, there is added to the kiln feed at least one reagent—the salt of an alkali metal. One or more additional reagents may also be added. This depends partly on the requirements of the particular earth being treated, and partly on the type of product desired. The flux-calcined product, unlike the straight-calcined, comes out a more or less snowy white.

After calcination, with or without flux, the kiln-burned material is re-milled and closely classified. This classification results in a minimum of three products: a waste grit, in part composed of low melting point components fused during calcination; at least one main powdered product; and a fine baghouse product. There may be two or even three of the main powdered products.

Granular products may be either uncalcined or calcined; most of them are calcined. Some earths are too soft to lend themselves economically to the manufacture of granules. Other earths, of intermediate hardness, make fine granular products, but calcination must be performed in chunk form in vertical shaft kilns to avoid high attrition losses. The particular granular product desired is made by crushing and screening after the earth has been hardened by calcination. Still other earths, notably from Nevada desert deposits, make good tough granules even in the natural state, and can be calcined after sizing. Granular products are used for insulation directly, or for insulating aggregates, for lightweight aggregates, for absorbent floor-sweep, for absorbent poultry litter, or for dehumidifiers, as in the case of De-Moist.

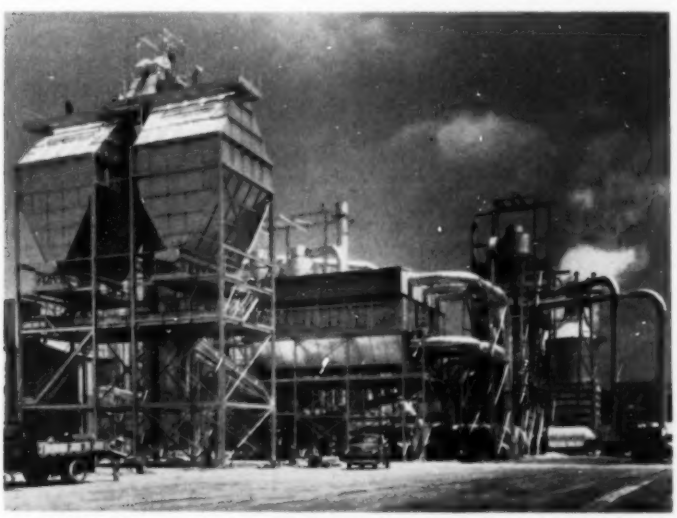
When the writer speaks of calcination, he is usually referring to calcination in the rotary kiln. However, granular products are calcined in either vertical shaft or rotary kiln. Calcined bricks, of course, necessitate special handling, and must be burned in a tunnel kiln, or similar device. In the past, a Herreshoff furnace has been used for calcining diatomaceous earth. There is no reason why a sintering machine could not be used to advantage, provided it is adapted to this finely-divided, lightweight and fluffy material. Flash calcination has been attempted, and may one day prove practical and economical. But, by and large, the rotary kiln, whose internal design has changed only imperceptibly for two generations or more, still ranks first as the calcining equipment giving the most consistent and dependable results.

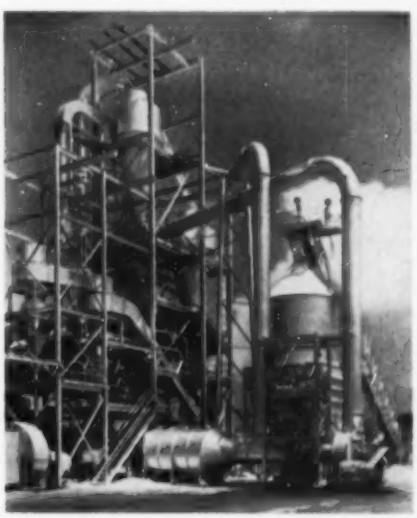
Men of character and ability are responsible for the great advance of the diatomaceous earth industry. That advance, in the last 50 years, has been almost phenomenal, whether considered from the standpoint of tonnage, of processing techniques, or of use applications. Fifty years ago, California's annual output of diatomaceous earth was about 400 tons; by 1910, it was 1,800 tons; by 1915, 12,000 tons; by 1920, 60,000 tons; by 1930, about 100,000 tons.

In the 1930's came the first major venture outside of California, with Dicalite's Terrebonne, Ore. plant. In the 1940's, Dicalite opened plants in Nevada and in Washington, and Eagle-Picher opened a plant in Nevada. Other enterprises of the 1940's were the Chick-Bed plant in Nevada and the Quincy Corporation plant in Washington.

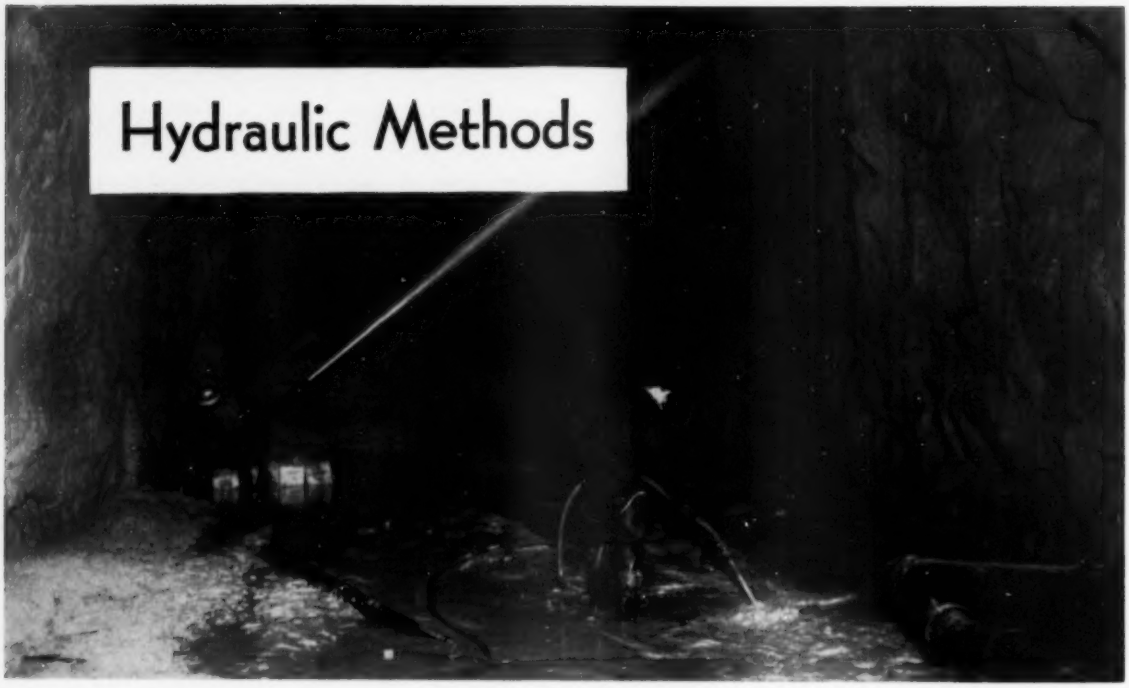
By 1940, annual U. S. production had reached 150,000 tons; by 1945, 200,000 tons; by 1948, 250,000 tons; today it has passed 300,000 tons, and is nearer 350,000.

In 1952, Great Lakes Carbon Corp.'s Dicalite Div. opened a large new, highly modern plant at Lompoc. Johns-Manville is completing a plant addition, also at Lompoc, that will increase their production facilities by an announced 40 pct. So the advances in tonnage and in process modernization have not slackened. We can expect still more and better-processed diatoms in the future. It seems past belief that the uses of the diatom can increase still further, and yet that, too, is happening. It is in fact an understatement to call this the non-metal of a thousand uses.





LEFT—View of newest, largest diatomaceous earth processing plant, the Lompoc, Calif. operation, Dicalite Div., Great Lakes Carbon Corp. RIGHT—Closeup of wet section of Dicalite's Lompoc plant showing crude elevator, storage bin, primary drying furnace, and small and large cyclone separators.



For Underground Mining of Silica Sand

by A. D. Bryant

APPROXIMATELY 2½ million tons of silica sand and ground silica with a value of $7.25 million is produced annually from the Ottawa, Illinois, district. These silica products come from the well known St. Peter sandstone formation which covers part of the Mississippi Basin extending from St. Paul on the north to the central part of Arkansas on the south and from Muncie, Ind., on the east to Kansas City on the west. Practically the whole state of Illinois is underlain by this formation except where removed by erosion.

Sand produced from the St. Peter sandstone in the Ottawa district is noted for its purity, uniformity, and quality. In addition to various screened grades for industry about 10 pct of the total production of the Ottawa district is ground into silica flour for use in glass, abrasives, enamel, porcelain, tile, for foundry and filter sand, and other uses.

Open-Pit Mining Methods

Where overburden thickness permits, open-pit methods are used in the district. The St. Peter sandstone is hard and homogeneous, but is loosely bound together by from 1 to 2 pct clay. After blasting this sand disintegrates so it may be readily pumped. The formation is therefore adaptable to hydraulic mining similar to gold mining operations. Waste dirt and rock are first stripped off and the sandstone blasted down by dynamite placed in vertical drill holes of 6 to 8 in. diam. The sand is then washed down by high-pressure hydraulic nozzles into a sump. A sand ejector or hydraulic elevator picks up the sand and delivers it to rubber-lined pumps which pump the material 2,000 ft horizontally and elevate it 130 ft to the top of the washing plant.

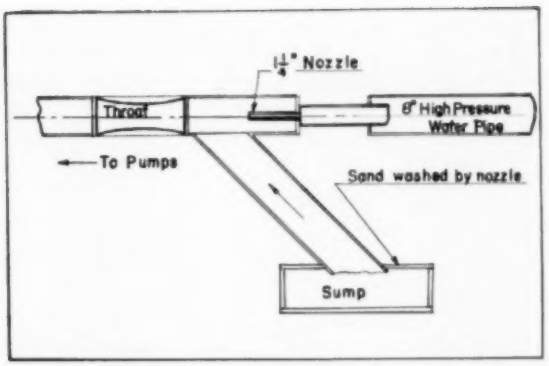
Clay is removed from the sand by two hydroseparators and the sand discharged into drain bins where the moisture is reduced to about 5 pct. The sand is then dried on steam coil dryers and screened into various grades.

Underground Mining Methods

Underground silica sand quarries have been operated by room and pillar methods for many years by the Pittsburgh Plate Glass Co. at Crystal City, Mo., the Pioneer Silica Co. at Pacific, Mo., the Silica Products Co. at Guion, Ark. and the Concrete Materials Co. at Clayton, Iowa. The methods at these four plants are practically the same and the rooms are usually 35 to 45 ft wide by 25 ft high. After blasting the sand is loaded by diesel or electric shovels and trucked to the processing plants.

Prior to starting Standard Silica Co.'s underground operation a careful study was made of the four underground operations mentioned. This investigation revealed no reason why underground methods could not be adopted to advantage in the Ottawa district. The sandstone at Standard Silica's plant in the Ottawa district is very easily drilled with the ordinary rock drill but it is homogeneous

A. D. BRYANT is Vice President of the Standard Silica Corp., Ottawa, Ill.



Sand ejector or hydraulic elevator used to transfer sand to pumps. Ejector shows in lower right of photograph on opposite page.

and there are comparatively few horizontal fractures. For this reason there is little difficulty in maintaining a good roof with rooms from 25 to 35 ft wide by 25 ft high. The fact that the sand disintegrates after blasting facilitates washing the material by hydraulic nozzles into a sump where the sand is picked up by sand ejectors or hydraulic elevators and conveyed through spiral-welded pipe lines to the rubber-lined pumps.

The operation is therefore very simple as the only equipment necessary to transport the sand from the face to pumps is high-pressure water, hydraulic nozzle, hydraulic elevator and pipe lines. There are no moving parts, and wear on pipe lines and the cast iron throat of the hydraulic elevator is negligible as compared to the wear and maintenance of trucks and shovels. Some expense is involved in extending the high-pressure water lines and sand lines but this does not amount to very much per ton of sand pumped.

Underground operation is carried on in the following cycle:

Three air drills are mounted on a jumbo equipped with caterpillar treads. A platform on this jumbo is raised and lowered by an electric hoist so the drills may be placed in the proper position on the 25-ft face. Two drillers operate three drills and drill approximately 52 holes 12 ft deep in 5 hr. The remaining 3 hr of an eight-hour shift is spent in changing the location of the jumbo and moving the machine in and out.

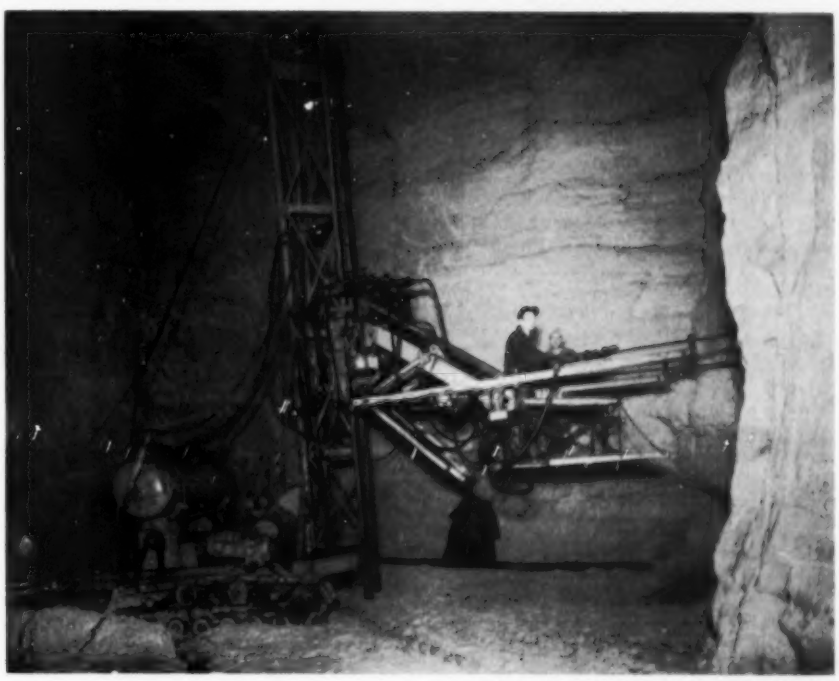
After drilling the two drillers load the holes with dynamite and blast in approximately 4 hr. Following the blasting three men from the regular open pit pumping operation come underground and pump out approximately 400 tons of sand which has been broken down. The time consumed in this operation is about 4 hr.

Surface vs Underground Methods

Sand is pumped from the surface quarry at a comparatively low cost for labor and explosives due to the nature of the formation. On the average 100 tons of sand will be pumped per man-shift in the open pit quarry which includes stripping 4 to 8 ft of cap rock, 2 to 4 ft of yellow sandstone which must be thrown away, well drill blasting, pumping, maintenance, etc. The cost of the underground operation compares favorably in output per man with the surface operation as no expenditure is made for stripping the overburden of cap rock and yellow sandstone. However, the underground operation is slightly more expensive as more dynamite per ton is consumed and the cost of blasting caps is higher than for the surface operation.

Equipment required for operating the underground quarry by hydraulic methods consists of the following:

1) Jumbo mounted on caterpillar treads and provided with air-cooled compressor, movable drill platform, 25-ft mast with three drills.
2) Hydraulic nozzle and hose connections operating at from 150 to 200-lb pressure.
3) Sand ejector or hydraulic elevator and pipe lines.
4) Vibrating screen to remove material on 4 mesh screen.
5) Rubber-lined sand pumps and pipe lines.



Jumbo mounted on caterpillar treads carries air compressor, movable platform, and 25-ft mast with three drills. Typical room is 25x25 ft.

Open Pit Forum

Trucks with Torque Converters

by Furman Byars

IT is not likely that Isbell Construction Co. ever bought a piece of equipment that was so freely discussed as the torque converter truck. Questions most frequently asked were: "How does this torque converter operate?" "What are the advantages of a torque converter?" "Why would anyone buy equipment with torque converters instead of conventional drives?" "How much will it cost to operate in comparison with conventional types?"

Experience gained by the Isbell Construction Co. using torque converter trucks in its stripping operation of a low grade copper ore body at Ray, Ariz., provides answers to some of these questions. Arizona is the land of sun and for many it serves as a wonderful winter haven, but that same sun means that for six or seven months of the year temperatures often reach 115°F and at night do not drop below 90°F.

The project involved removal of approximately 45 million tons of waste and ore in stripping and developing an open pit copper mine for Kennecott Copper Corp. Trucks operated three shifts a day and, with few exceptions, seven days a week. The locality is very dry and the dust problem is acute. Hauls were comparatively long; up to 2½ miles one way with a vertical lift of material from pit to dump of 300 ft. Trucks pulling on long hauls against adverse grades, under extreme heat and operating the required number of hours per month developed many problems never previously considered.

Equipment

Remarks are limited to the Allison Torque Converter because it is the only type used. Torque converters and hydraulic transmissions at the Ray operation were used in a Euclid rear dump truck, model FFD, having a factory rated capacity of 34 tons net. The dump body is mounted on three axles carried by 16:00x25 tires. The truck is more or less novel in its design, in that it is powered by two diesel engines rated at 190 or 200 hp each driving separate rear axles. The truck has two engines, two torque converters, two transmissions, two drive lines and two differentials. The diesel engines are General Motors model 6-71 and Cummins model NH; the torque converters and transmissions are made by Allison.

The truck has no clutch. The driver sets the truck in motion by moving a selector lever which opens or closes a hydraulic valve. Through this valve, hydraulic pressure is applied to disc type clutches in a planetary transmission, which has three speeds forward and one reverse. Speed is controlled by selection of gear ratio and throttle position. Selection of any of the desired speeds can be made under full load conditions without any perceptible loss of vehicle speed. The transmission is a constant mesh type with no sliding gears and can be shifted under full throttle.

FURMAN BYARS is Superintendent, Isbell Construction Co.

Torque converter equipped trucks at the Ray Pit are powered by two diesel engines rated at 190 or 200 hp each.

Torque Converters

A torque converter multiplies the torque of the engine by transfer of oil within the converter. The converter is charged with oil under pressure by an outside supply pump. The torque converter has four main parts; a pump attached to the engine, a turbine attached to the output shaft, a first stator and a second stator which are mounted as an over riding clutch and in only one direction.

The easiest way to describe the operation of a torque converter is by showing what happens from standing start until the vehicle reaches normal speed. When the throttle is opened and the engine approaches torque peak, oil in the pump is moved with centrifugal momentum and circulates through the vanes of the now stationary turbine. As the oil is altered in direction by the curvature of the turbine blades it asserts a direct force upon the turbine. The oil, upon leaving the turbine, is directed to the first stator and thence to the second stator. Since the stator blades re-direct the oil back to the converter pump torque multiplication exists. As the speed of the output shaft increases the discharge of oil from the turbine is directed on the back side of the first stator and then to the second stator and they both become free wheeling. When this point is reached the engine shaft and converter output shaft turn at almost the same rpm. Torque is no longer multiplied and the converter operates as fluid coupling.

The reason foremost in everyone's mind for buying torque converter equipment is reduction of errors caused by the human element in driving. An automotive diesel generally has a usable speed range of 1500 to 1900 rpm. A converter increases this speed range from 0 to 1900 rpm and at the same time torque multiplication is obtained, thereby eliminating the necessity for frequent shifting. Since there is no mechanical connection between the engine and the converter output shaft, the engine

does not stall nor does it lug or slow down to injurious speeds. Shock loads are greatly reduced and the truck is propelled more smoothly.

One of the advantages of this truck is its relatively low labor cost. One of the best ways to whip the ever increasing cost of labor is to effect greater production per man. The 35 ton capacity of this truck is certainly a help.

Comparative Performance

Never having operated a truck of this size with a conventional drive it is difficult to properly compare performance, since this truck has practically a double set of component parts. However, it can be compared with a truck having the same engine and differential, but with conventional clutch and transmission, such as Euclid's model 49FD.

On similar hauls, measuring 1.4 miles each way over an average grade of 5 pct, the 49FD carrying 17.5 tons moved 66 tons per hr, while the FFD carrying 41 tons moved 152 tons per hr. If we double the single unit we get 132 tons per hr, or 20 tons per hr less than attained by the FFD truck.

Similarly, if the performance of 25 ton truck, which is a two axle unit powered by 300 hp engine, is compared, it is found that it moved 91 tons per hr on the same haul. Incidentally, this smaller truck is very satisfactory and its performance is used as a criterion for comparison with other trucks.

It may be surprising that Isbell uses a load factor for the FFD of 41 tons when the factory rated capacity is 34 tons. From observation at Chino and Ray, it was decided to experiment with larger, 30 cu yd, loads compared with regular, 24 cu yd, loads over the same road. The haul road was broken into sections at grade changes and several tachometer and speedometer readings were taken in each section for both 24 and 30 cu yd loads for an accurate check of variation in cycle time or performance. This experiment was conducted on a haul approximately a mile long with an average grade of 5 pct. The results showed that although the loading time was increased 30 seconds the overall cycle time increased only a minute and 10 seconds and that production was increased from 142 tons per hour to 163 tons per hr.

To further observe the effects of this load increase upon engines, they were equipped with thermocouples to accurately determine temperatures. By plotting the exhaust temperature readings on a graph at 30 second intervals during the cycling time it became apparent that the engine under full throttle worked no harder with 41 tons than it did with 34 tons or even 25 tons. Obviously then, the real problem is balancing torque converters to loads and subsequently engines to torque converters.

Before analysis of mechanical problems and difficulties experience may show how changes brought additional trouble free operation. Ray is removed from a manufacturing center, outside personal touch is somewhat lost, and many experiments were completed through the initiative of local shop personnel.

From the time the trucks were delivered until July 1, 1950 there were sad experiences, mainly with engines. By the end of July, the factory, together with shop personnel, had incorporated changes for later designs. During that period the trucks were available 79 pct of possible time. Since July, 1950 the available truck time has increased to 85½ pct. During the time of engine troubles the engine overhaul period averaged 2040 hr. Since engine trouble has been largely overcome the overhaul period has been increased to 5400 hr. Increase in availability cannot be credited solely to engines because during that same period many changes were incorporated in torque converters and transmissions. Many changes were developed in the local shop and experiments are being continued to increase the period between breakdowns.

Hauls are up to 2½ miles one way with a vertical lift from pit to dump of 300 ft. Torque converters helped solve many problems never previously considered.

Some of these trucks have operated as much as 14,000 hr. Sixty five torque converters have been overhauled, for an average life of 2400 hr and 21 transmissions for an average life of 7200 hr. The average cost for parts used in the overhaul of a converter is $125.

There has been very little mechanical trouble with the torque converters by leakage of oil from seals. The greatest contributing factor is the outside charging pump. Its life has, at times, been very short. When the pump does break down it sometimes occurs suddenly and without warning. The charging pressure to the converter is immediately lowered, and the converter becomes overheated. The excessive heat may last for only a short time but this is enough to destroy seals.

Transmission problems have been few, but acute. The ring gears of the planet drive were found to be too soft resulting in clutch plates riding and wearing the gears. No trouble has been experienced since the manufacturer nitrited the gears. The bypass valve on the transmission, designed to protect the system against high pressures, was found to be inconsistent. It was dumping oil at too low pressure, causing injury to the torque converter. This was replaced with a by pass valve of local design. It was necessary to change the method of taking the thrust on the output flange and also to hold the main shaft solid instead of allowing it to float. When replacing some of the major parts overhaul costs were about $560.00 or 8¢ per operating hour.

It has been felt that the engine might be too low powered to pull the torque converter at a safe speed. The stall of the converter is at 1700 rpm motor speed. In other words, the safe speed of the motor with the output shaft of converter held stationary is 1700 rpm and the truck is working on almost all grades at an average of about 1750 rpm. It has been contended that the engines work at too much of a constant load and if run at 1850 to 1900 rpm might last longer. However, considering the time between overhauls, approximately 5400 hr, and the cost of overhauls, the theory may be wrong. Engines developing approximately 15 percent more horsepower are now being tested.

Future of the Pennsylvania Anthracite Industry

by Alvin Kaufman

All opinions expressed in this paper are those of the author, and do not necessarily represent those of the State Planning Board, Pennsylvania Department of Commerce.

ANTHRACITE mining in the United States is concentrated in a 480 sq mile section of northeastern Pennsylvania. The producing area is broken up into four fields, known as the Northern, Southern, Eastern Middle and Western Middle fields. These areas represent intricately folded coal measures in synclinal residuals left by erosional forces.

Destruction by erosion has eliminated some 95 pct of the original coal. However, discounting the 5 billion tons removed to date, 8 billion tons are believed to be still recoverable.

The intense folding of the area which was responsible for the formation of anthracite was also responsible for the present difficulties encountered in mining, and for the high percentage of fines which are produced.

For the last three decades anthracite mines have been among the few mineral industries to show a decline in production, as illustrated in Fig. 1. This has been due to the competition of oil and gas for the home heating market. This decrease has led many persons to believe that anthracite is as outmoded as the kerosene lamp of yesteryear. However, it is doubtful if any industry producing 40 million tons of material valued at $360 million is going to pass away within the next few years. It is quite true that hard coal was hurt and hurt badly by the many wounds which have been inflicted upon it. However, despite competition from oil and gas, and despite its hitherto poor marketing set-up the industry is starting to walk again. It has a long way to go to reach good health, but it is struggling toward its goal.

Mining Problems

Problems besetting the industry are unique, for here is a fuel which is cheaper than most of its competitors, which has ample reserves, which is smokeless, which yields an intense heat, and yet has been unable to hold its own. If the industry is to improve its present position, it is necessary that it increase the mechanization of its mines, conquer the severe water seepage problem, as well as find a use for the larger tonnages of fine-sized materials produced.

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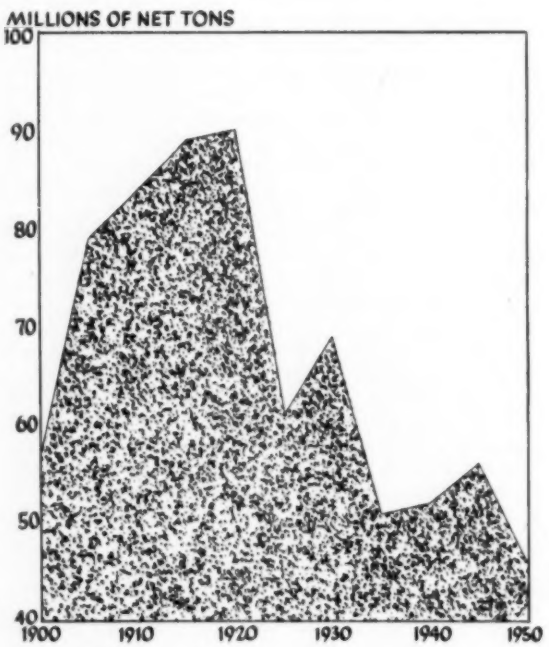

Fig. 1—From this comparison it can be easily seen that anthracite coal production reached its peak in 1920 and has been in a more or less steady decline since then.

It will be noted in Fig. 2 that the tons produced per man day have shown but a moderate increase in the last 20 years. This, despite the fact that 27 pct of 1950 production was from stripping operations. It would thus seem that technological advancement in the anthracite area has done little more than balance increasing difficulty of extraction.

Lack of mechanization coupled with the mining of thinner and more severely pitching seams has resulted in the downward productivity trend, and in constantly rising costs. If the industry is to survive in the home heating market, the lack of mechanization must be corrected. This fact has been recognized by the anthracite producers for many years. However, they have been powerless to correct the situation because of the steeply pitching and seriously faulted seams which would have to be worked by mechanical mining devices. The larger manufacturers of these devices have been unwilling to incur the expense of developing new machines for anthra-

cite due to the small prospective demand, and the keen competition presently existing within the bituminous mechanical mining machine industry. The smaller machine manufacturers have been willing, but have lacked the capital. As a result, the majority of machines used in the hard coal mines are those adapted from the soft coal mines.

In an attempt to alleviate this situation, the Bureau of Mines has stepped into the picture by building the Anthracite Research Laboratory at Schuylkill Haven, Pennsylvania. J. B. Buch, supervising engineer of the laboratory, recently described the development of a scraper-shaker-loader for driving gangways into seams with steeply pitching beds. This machine would permit the loading of material from a full round into a single trip of cars at a maximum loading rate of 150 tons per hr. One and one-half hr is required for a round, with time remaining within a work shift for drilling, blasting and timbering. The net result of the use of this machine would be to decrease the number of shifts required to complete a mining cycle from two to one. The Bureau is also working on the improvement and adaptation of a light weight German shearing machine for use in both flat and pitching beds; work is being pushed on a vibrating blade coal planer, as well as a telescoping tunnel-shield type face support. The latter will keep timbering time in step with that needed for actual mining. As the bugs extant in the utilization of all new machines are ironed out, and as these machines gain more widespread usage, production per man should increase and cost per ton decrease. Fig. 3 illustrates difference between bituminous and anthracite cut mechanically.

Due to high cost of underground labor and lower productivity per man, present mining trend is in the direction of strip mining. The future will undoubtedly see larger and deeper strip operations having a longer life. These operations will require a large capital outlay, as well as accurate engineering planning. However, it is doubtful if the majority of anthracite produced will ever be mined from strip beds due to near exhaustion of seams close to the surface.

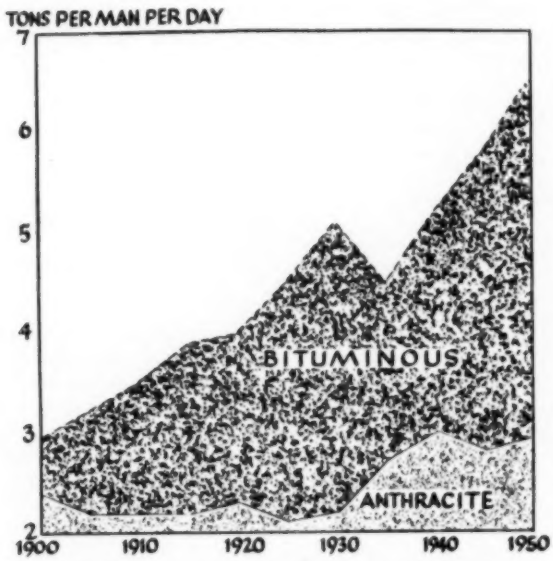

Fig. 2—Anthracite tons per man day have shown only a moderate increase in the last 20 years, despite the fact that 27 pct of 1950 production was from strip operations. Rise of bituminous production is almost spectacular.

The hazards and costs of mining anthracite have been constantly aggravated by the seepage of water into the mines. As the workings penetrate deeper into the earth more than the present average of 19 tons of water per ton of coal mined will have to be removed. It has been estimated that at least 25¢ must be added to the cost of each ton of coal mined because of this water disposal problem. Part of this difficulty is due to the denuding of forests that once retained excess precipitation. There have been times when the equivalent of as much as 89 pct of the rain water has been pumped from the mines. To combat this problem, reforestation has been undertaken, and new pumps capable of handling greater volumes of water at lower per gallon cost are being developed. In conjunction with the new pumps it has been suggested that it might be advantageous to impound the ground water in underground pools. These pools would serve to maintain the water table at a constant level and would be so situated that they would act as a drainage basin for the mines. It might be possible to employ water pumped to the surface for breaker use, dust control, hydraulic back filling, combatting mine fires, and to aid in the transportation of material in gently dipping places. Inasmuch as a large amount of the water handled would be highly acidic it probably would be necessary to treat it with lime in order to protect equipment, and thus lower cost of materials and maintenance.

Future Utilization

Throughout its long history, anthracite has always been used as a fuel. For this, our grandchildren will probably curse us. In anthracite we have a material containing from 92 to 98 pct pure carbon. Here is one of the richest sources of organic compounds known to mankind. Yet, to date, millions of tons are destroyed annually because we lack the technology to solve the secret of these black diamonds.

Today, anthracite is used predominately as a domestic fuel. This, in itself, presents a problem, for the traditional hand-fired furnace burns the larger or domestic sizes, that is, from pea on up to broken; while 40 pct or more of the mined production, due to geologic conditions, is in the steam sizes ranging from buckwheat No. 1 on down through rice. Thus, the mine operators are faced with the problem of selling the finer sizes of coal at a loss or storing them. Before 1947 much of this material was dumped into the rivers. However, in recent years Pennsylvania law has required the desilting of breaker water. The effect of this requirement is best shown by the comparison of the silt load of 349,000 tons dumped into the Schuylkill River in November 1947 with the March 1949 figure of 7000 tons.

The problem involved in the high proportion of the fine sizes is pointed up when one considers that the average sales realization for the domestic sizes is $11.27 while the steam sizes have a realization of $4.79. Production cost is estimated at $8.70 per ton. It will be seen from these figures that the profits made on the larger sizes of coal must be used to balance the loss of the smaller sizes of coal. As a result, many operators show a loss at the end of each year.

Anthracite sales, it is hoped, will change as the new automatic burners gain in popularity. These operate on a new coal burning principle, and may be adapted for steam, hot water and vapor heat.

Most of these units are stoker fed and utilize the smaller sizes of coal. The comparative heating cost of anthracite coal is much less than the other competitive fuels.

Table I. Retail Cash Fuel Prices, June 15, 1952

City	Chestnut	#1 Buckwheat	#2 Oil* Equivalent per Ton
Philadelphia	$20.65	$15.40	$24.58
New York	21.75	14.40	25.39
Boston	24.95	18.00	25.39
Baltimore	21.50	15.50	24.58

*201.5 gallons #2 oil = 1 ton (University of Illinois).

Anthracite has been outsold because the dealer has insisted on selling coal rather than service. If anthracite is to remain in the home market, it must supply ash removal service, degree-day delivery, a service and repair contract, and budget payments.

Present emphasis on smoke control will probably expand the commercial fuel market for anthracite. An anthracite-bituminous mixture eliminates the need for a big combustion space, for high boiler settings, for constant agitation of the fuel bed, and the necessity of adding large quantities of oxygen. Anthracite gives a higher boiler efficiency as well as a reduction in fuel consumption, smoke, and labor.

A huge future market is seen in the utilization of hard coal for the production of electricity.

Large quantities of fines are now stored in the anthracite area from the days when nothing under pea size could be used as a fuel in a hand fired furnace. These culm banks are now being utilized by several electric companies for power production. One of these plants will have an eventual capacity of 550,000 kw-hr. Present consumption is 1.1 pounds of fine sized anthracite per 11,000 Btu produced.

Anthracite makes an excellent sintering medium for eastern magnetite as well as being useful in improving the yield and strength of metallurgical coke. Hard coal is also the standard fuel for zinc smelting.

Notwithstanding these advances in its use as a metallurgical fuel, it is believed that anthracite will some day be more desirable as a non-fuel. To date it has found widespread usage as a filter medium in water settling basins. It is easier to flush clean and retains its adsorptive capacity longer than sand. This use will undoubtedly expand through the years.

Experiments conducted at the Pennsylvania State College have shown that a more profitable utilization can be obtained from sulphur dioxide, a by-product of zinc production, than the present usage for the manufacture of sulphuric acid. These experiments have shown that carbon di-sulphide can be produced from SO_2 with a cheap source of carbon. These same tests have revealed anthracite to be the best source of this carbon because of its great initial reactivity and because it is continually reactivated by the reactions within the process. Inasmuch as Pennsylvania is the largest zinc smelting state in the country, it would seem that the carbon di-sulphide industry would be a natural for the Commonwealth.

Another avenue of endeavor for anthracite would seem to lie in the direction of gas generation. Buckwheat and rice anthracite are excellent for this purpose because they do not produce tar or soot and they have a high ash-fusion temperature. These sizes mixed with oxygen in a Galusha producer, give a good gas at an estimated cost that is slightly higher, at present, than that of natural gas. The fluid thus produced could serve as the basis for hundreds of other compounds, particularly the production of ammonia and methanol. In addition, water gas for domestic heating purposes could be produced, as well as producer gas. The latter is presently being produced from anthracite at Trenton, N. J.; Baltimore; and Norris, Tenn. It is used in such varied processes as the annealing of steel castings, biscuit baking, and the heating of ceramic kilns. Perhaps this is the key to unlock the door to the chemical riches anthracite contains.

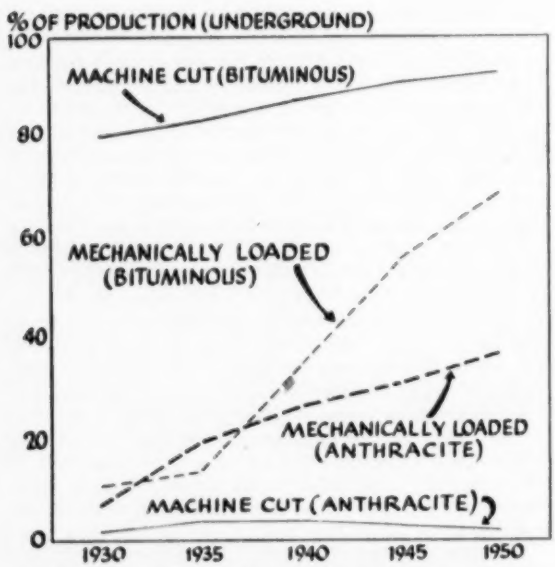

Fig. 3—Difference in mechanization between bituminous and anthracite mines seems to be great. Percentages of the two types of coal mechanically cut differs widely.

Table II. Composition of Producer Gas

	Anthracite Pct by Volume	Bituminous Pct by Volume
CO	42.4	27.0
CO_2	17.2	4.5
H	37.7	14.0
N	1.8	50.9
O_2	0.2	0.6
Methane	0.8	3.0
Btu per cu ft	266	150

Conclusion

It would seem to the author that the future of anthracite will be as a raw material, rather than as a fuel. However, its fuel sales must be increased to finance the research work being carried on by the Anthracite Institute, and other organizations.

As chemicals from hard coal come of age it will no longer be necessary to produce anthracite in larger sizes, and inasmuch as geologic conditions are more suitable for production of fines, costs should decline.

Once the difficult technical problems have been solved which beset hard coal, we shall discover that the composition of oil and gas has proved to be a boon. Because of it we shall have opened vast new vistas of opportunity.

Fig. 1—Northampton quarry, operated by Universal Atlas Cement Co. This laminated rock is easily drilled, blasted, and crushed.

Cement Rock Beneficiation at the Universal Atlas Cement Co., Northampton, Pa.

by L. J. Boucher

The beneficiation process at Northampton is described and reasons are given for installing a flotation plant. The economics of running the plant, the difficulties of operation, and subsequent remedial action are discussed.

CEMENT rock from a quarry operated by the Northampton plant[1,2] of the Universal Atlas Cement Co. has a relatively low average $CaCO_3$ content. To meet specifications required for cement manufacture it was formerly necessary, over succeeding years, to add increasingly larger amounts of high-lime, low-alumina rock, which was obtained from a quarry operated by the company at a point along the New Jersey-New York state border, 79 miles from Northampton. Previous to 1940, 8 pct of total kiln feed consisted of high-lime rock. From 1940 to 1943, this figure jumped to 13 pct. Today it would be necessary to use 18 pct of this high-lime to meet current specifications. Increasing transportation costs made this an expensive operation. In addition, this high-lime rock quarry was contaminated with granite and high magnesia pockets.

As there was no available high-lime rock in the Lehigh Valley, it was decided in 1936 to install a pilot plant where experimental work was done on processing local cement rock with flotation and burning the resulting product in a small kiln. This plant was in operation for six months and at the end of that period a suitable method of flotation had been established. From these studies it was also found that if the proportion of shipped-in rock rose slightly above 10 pct it became economically advisable to use the flotation process. It is the object of this paper to present briefly the operation of the flotation system at the Northampton plant. It seems advisable to comment on the raw material used in this process.

The Northampton quarry lies geologically in what is known as the Jacksonburg formation of Trenton limestone.[3] In Pennsylvania this extends in a southwesterly direction from Portland, through Nazareth, Bath, and Northampton and across the Lehigh River to Ormrod. This argillaceous limestone was formed from marine deposits laid down during the early Ordivician period when a shallow sea covered most of Pennsylvania. Carbonaceous matter in the original deposit formed graphitic carbon under later metamorphic forces. From this graphite the cement rock of the Lehigh Valley derives its dark color. Most of the graphite must be removed before lime flotation can proceed.

The quartz in this rock is desirable from the chemist's point of view, as it furnishes him with a form of silica necessary in the calcining process, but its abrasive properties damage pumps and piping. Aluminum silicates of potassium, sodium, iron, and magnesium are found in the form of sericite. Calcite averages from 69 to 76 pct. Iron in the form of iron pyrites is also present and has no deleterious effect. Talc is present in minute quantities. The

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The Transactions papers appearing in Mining Engineering during 1953 will subsequently appear in AIME Transactions Vol. 196, and may be permanently referenced to that volume.

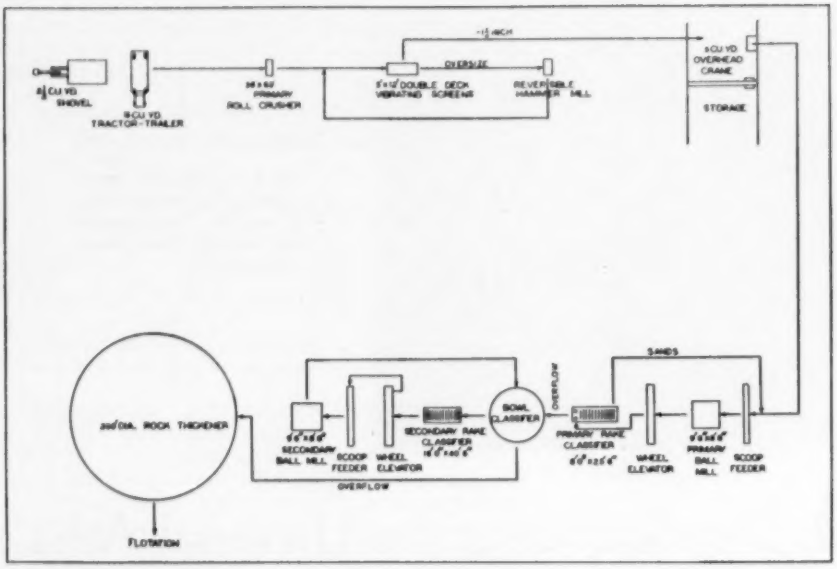

Fig. 2—Crushing and raw grinding flowsheet from loading to thickening stage. Slurry from the secondary mill is 90 pct —200 mesh.

mineral composition of Northampton rock is shown in Table I.

The rock has been deposited in laminations, see Fig. 1, and it is therefore very easy to drill, blast, and crush. It is loaded with a crawler-type electric-shovel 2½ cu yd dipper, Figs. 1 and 2, and transported from the quarry by means of truck trailer to a 36x60-in. single-roll crusher, giving a —8-in. product, from which it passes through doubledeck scalping 1¼-in. screens. The oversize passes through a reversible hammermill from which it is returned to the scalping screens. The stone passing through the screens goes to storage and into raw rock grinding, which because of flotation must be done by wet process. Primary grinding is a closed circuit consisting of a 9 ft 6 in. x 8 ft 8 in. ball mill, with 4-in. grinding media and an 8 ft x 25 ft 6 in. Dorr multizone classifier with 2½ in. per ft slope. The fines are discharged into a 27 ft 0 in. diam bowl classifier, forming a closed secondary grinding circuit, with a 9 ft 6 in. x 8 ft 8 in. ball mill with 1½-in. grinding media,[4] see Figs. 3 and 4. This circuit delivers an end product of which 90 pct passes a 200-mesh screen. This slurry, containing 25 pct solids, is then pumped into a 200-ft thickener, see Figs. 6 and 7, from which it is drawn at an average 54 pct solids and pumped into a small mixing tank, where the slurry is diluted to 35 pct solids and is then stored in a turbomixer.

Table I. Composition of Northampton Rock

Mineral	Pct
Calcite and dolomite	75.00
Quartz	8.50
Iron pyrites	.56
Graphite	.50
Sericite	15.44

The next step in the process is a separation at 30 microns in a continuous solid-bowl type of centrifuge, 54x70 in.,[5] see Fig. 8. A peculiar situation arises when separation takes place in the centrifuge. As shown in Table II, the centrifuge feed will contain 73.17 pct $CaCO_3$, the cake or +30 micron fraction will contain 75.46 pct $CaCO_3$, and the effluent or —30 micron fraction will contain 71.02 pct $CaCO_3$. In any grinding, the harder particles will be the coarser, and since calcite and quartz are the hardest minerals present as compared with sericite, the centrifuge cake will contain a larger amount of $CaCO_3$ and SiO_2.

The outer bowl of each centrifuge, see Fig. 8, rotates at an average speed of 320 rpm, while the inner bowl rotates at 304 rpm, maintaining a 19 to 20 speed ratio with the outer bowl. At this speed a 30-micron separation takes place. Naturally, a variation in speed will change the dividing line between particle size separation. It is also important that the specific gravity of feed be set at 35 pct solids, as a more viscous mixture results in poorer separation due to difficulties of removing the smaller particles with insufficient water.

The cake with the 75.46 pct $CaCO_3$ is stored for blending without further processing, as the flotation process will furnish a product of much higher lime concentration, and the blending with other products results in a kiln feed of required composition.

Each centrifuge produces on an average 56.2 tons per hr if its separation is efficient. Presented in Table III are the cumulative percents finer than micron sizes of the feed and the discharges.

This operation is continuous, except on weekends, when the whole flotation system is shut down. Before stopping, it is necessary to wash and remove any deposits, which harden if left standing, causing an unbalanced load and considerable vibration when the centrifuge is started. The main difficulty experienced in this phase of the operation was that in the cone on which the screw was mounted and into which the feed pipe discharges the rinse water did not drain completely. A number of 1-in. diam holes were drilled along the circumference of the cone to correct this condition. There has been no trouble since caused by vibration.

After the effluent leaves the centrifuge, it is diluted to 17 pct solids and fed into a turbomixer, from which it is pumped into a constant head feed tank for the flotation cells. The flotation system, see Fig. 9, consists of four parallel rows of eighteen 66x66x27 in. cells.[6,7] These are of the mechanical-agitation type, each cell being equipped with a motor-driven vertical rotor. The most efficient speed of these rotors was found to be at 420 rpm.

Fig. 3—Grinding circuits utilize multizone classifiers.

Fig. 4—Fines overflow 27-ft bowl classifier to rock thickener.

Skimmers placed along the sides of the cell group accelerate the froth removal. Average production rate, per row of 18 cells, is 10⅜ tons of solids per hr, or 0.148 tons per hr per cu ft of cell volume. Power requirement is 4.5 hp per cell. There is very little maintenance, as both stator and rotor bars are rubber-covered and require little attention.

In operation, a carbonaceous concentrate is taken off in the first three cells of each row. This is relatively low in lime and quite high in micaceous and clay materials. The object of eliminating this material at this point is to facilitate the ensuing flotation of lime and to save on the consumption of reagents which would be absorbed by this carbonaceous material. Diesel oil is used as the collector and methyl isobutyl carbinol as the frother.

Table II. Analysis of Centrifuge Feed, Wt Pct

Compounds	Feed	Cake, +30 Micron	Effluent, −30 Micron
SiO_2	14.28	13.80	14.80
Al_2O_3	4.98	3.86	5.96
Fe_2O_3	1.60	1.50	1.68
$CaCO_3$	73.17	75.46	71.02
$MgCO_3$	4.42	4.52	4.23

Normally the amount of collector reagents added is held constant and the frother is varied to maintain the carbonaceous concentrate at 58 to 60 pct calcium carbonate at the first three cells. If this carbonaceous froth goes below 58 pct $CaCO_3$, the amount of carbon removed is too low. If it goes above 60 pct, too much $CaCO_3$ is being removed, and this will be reflected in a lower efficiency. Samples are tested regularly, for care must be taken not to draw off calcium carbonate excessively. The carbon fraction joins with the final rejects and is discarded.

Table III. Centrifuge Performance, Cumulative Percents Finer Than Micron Sizes of Feed and Discharges

Micron	Feed	Cake	Effluent
44	88.7	83.7	100
30	72.2	64.3	100
20	64.9	53.2	94.6
15	57.8	42.7	93.3
10	48.8	34.5	83.4
8	45.2	31.7	77.4
7	43.1	30.8	73.2
6	40.6	29.7	69.8
5	38.3	28.9	65.9
4	35.5	27.8	60.2
3	34.7	27.0	55.0
2	29.6	25.5	48.8

In the fourth cell, the beginning of the high lime concentrate separation, calcium lignin sulphonate is added to disperse the remaining carbon. Anionic reagents, oleic acid, and saponified tall oil are used from this point on, saponified tall oil in Cells 4 to 14, and oleic acid in Cells 4 to 18. Thus oleic acid alone is used in the last four cells, because it is the better collector and is required when the recover-

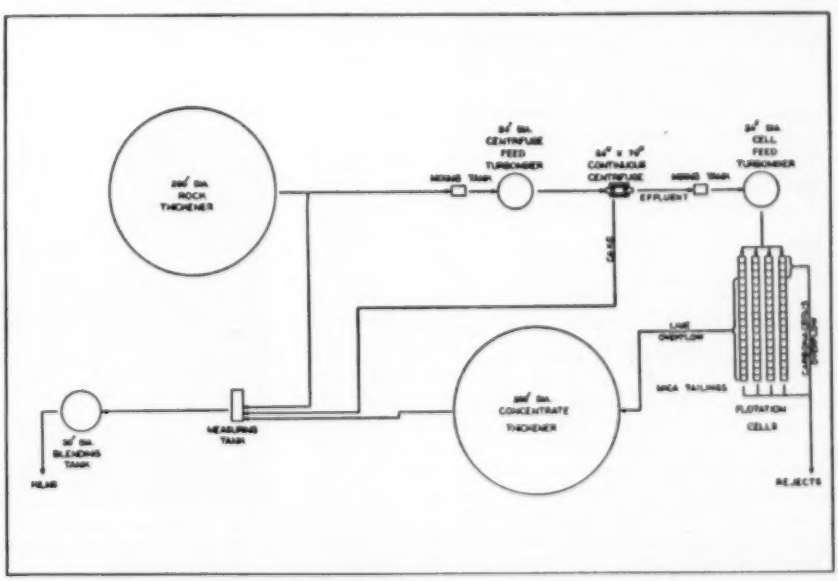

Fig. 5—Slurry is drawn from a 200-ft rock thickener at an average 54 pct solids. In the mixing tank it is diluted to 35 pct solids, after which it is stored in a turbomixer.

Fig. 6—The 200-ft thickener receives rock slurry from the grinding circuit. In left foreground is a small clear water overflow tank.

able calcite is as low as it is in these cells. As tall oil is the less expensive of the two, a ratio of 71 pct tall oil to 29 pct oleic acid is used overall. Reagent regulation requires careful adjustment, see Table IV.

These anionic reagents react with strong cationic minerals, such as calcite, at the surfaces of the particles, while with the other minerals present there is no such reaction. This reaction forms a compound which is insoluble in water and buoyant, rendering the particles water-repellent. Myriads of tiny air bubbles are whipped throughout the cells by me-

Table IV. Regulation of Reagent Consumption

Reagent Consumption	High Lime Concentrate, Lbs Per Ton
Diesel oil	.161
Methyl isobutyl carbinol	.231
Oleic acid	.314
Saponified tall oil	.793
Calcium lignin sulphonate	.953

chanical agitation of the rotor, so that the air bubbles attach themselves to water-repellent particles and raise them to the surface, see Fig. 10. Normally the bubbles burst at the surface and drop their mineral load, but a froth is maintained at the surface primarily by addition of the frother, methyl isobutyl carbinol, in the carbon cells holding the mineral until a skimmer removes the froth. A pH regulation of reagents is not necessary. Limestone slurries are normally at the optimum pH for best separation, which is about 8.2.

Flotation of cement rock is affected particularly by temperature, especially in winter, by grade of rock, variations in fineness, and amount of solids. In the case of exceptional changes in the grade of rock, changes in the amounts of collectors must be

Table V. Analyses of Flotation Plant Products

Compound	Cell Feed	High-Lime Concentrate	Carbon Rejects	Mica Rejects*	Combined Rejects
SiO_2	14.80	6.72	15.36	47.24	39.32
Al_2O_3	5.96	3.00	6.96	15.73	13.59
Fe_2O_3	1.68	1.30	4.00	2.68	2.43
$CaCO_3$	71.02	84.06	59.87	21.61	31.77
$MgCO_3$	4.23	3.27	5.14	7.31	7.13

* Mica rejects are the underflow from the last cell.

made. The percentage of $CaCO_3$ in the lime concentrate froth can be increased by decreasing the amount of frother or collector. This, of course, decreases total volume of concentrate derived and increases the amount of rejects per ton of cell feed. Calcium carbonate titrations are made regularly on samples of high-lime concentrate from each cell and on underflow rejects from the last cell.

The high-lime concentrate taken off Cells 4 to 18 inclusive is slightly above 84 pct $CaCO_3$ and this

Fig. 7—Raw rock thickener, left; flotation cell house and blending tanks, center; and 200-ft concentrate thickener, right.

flows by gravity into the high-lime 200-ft concentrate thickener which serves as a storage tank and water remover. The carbonaceous concentrate taken off Cells 1 to 3 combines with the underflow at the end of the cell row and is discarded by pumping to an abandoned quarry. Overflow water from the high-lime concentrate thickener is retrieved for use again in the flotation system.

Representative analyses of the flotation plant products will be found in Table V.

Data of a typical day's operation, using average-grade quarry rock, is as follows. On a dry tonnage basis, 2680 tons of cement rock are fed to the two centrifuges, of which 1340 tons are of the +30 micron fraction in the centrifuge cake, and the other 1340 tons of −30 micron fraction in the centrifuge effluent that constitutes cell feed. Of this latter 1340 tons, 1008 tons are high-lime concentrate at 84 pct $CaCO_3$ and the remaining 332 tons are rejects at 31.77 pct $CaCO_3$. Of the total volume of rock fed to the centrifuges, 87.6 pct $CaCO_3$ is recovered by the centrifuges and cells. Of the total volume of rock fed to the cells, 75.2 pct $CaCO_3$ is recovered. Varying with the quality of rock, see Fig. 8, an average 33.8 pct of cement rock is drawn directly to blend tanks for blending along with centrifuge cake and high-lime concentrate. With these materials available at all times, together with sand and pyrite cinders, production of any type of cement can be started in a short time. In every case these materials can be proportioned to give satisfactory kiln efficiency and make cements meeting all requirements of all specifications and consumer demand. If only one kind of cement is made and rock from thickener is quarry average rock, the amount of rejects discarded is 8.2 pct of total rock quarried. When one kind of cement only is made, if grade of

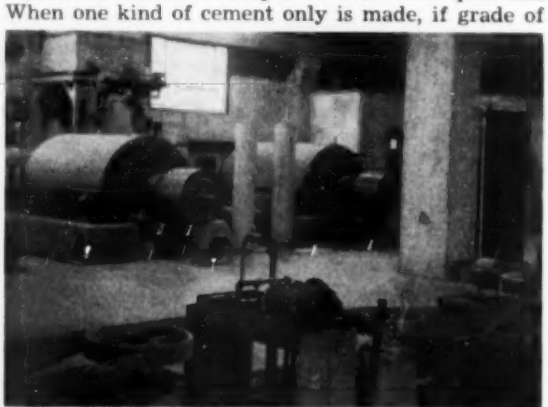

Fig. 8—The outer bowl of each continuous 54x70-in. centrifuge rotates at 320 rpm, the inner bowl at 304 rpm, effecting a 30-micron separation.

Fig. 9—A view of the flotation system, four parallel rows of 18 cells each. Mechanical-agitation cells are 66x66x27 in.

Fig. 10—Air bubbles whipped throughout the cells attach to water-repellent particles and rise with them to the surface.

rock is above 73.2 pct $CaCO_3$, the system's efficiency improves proportionately. If other types of cement are made or the grade of rock falls below 73.2 pct, the percentage of rejects increases.

Personnel required for one 8-hr shift includes one operator and one helper, one pumpman, one pumpman helper, and one laboratory control man. These men handle centrifuge operation, flotation, blending, and pumping to kilns. Flotation normally operates 24 hr per day, 5 days per week. When high-lime concentrate thickener becomes full, flotation operations must be curtailed. Grade of rock, amount and type of cements made, and kiln consumption are the governing factors.

This entire operation, centrifuging and flotation, occurs in a building known as the Cell House, in the basement of which flotation reagents are prepared in six make-up tanks. These tanks are complete with steam coils and agitating apparatus. The Cell House contains its own oil-fired heating plant for this purpose.

As previously stated, it is very important to control the density of material to centrifuges. Formerly this was done by weighing slurry samples from the mixing tanks every half hour and adjusting the water flow accordingly. As the specific gravity of the thickener slurry varied from 40 to 60 pct, with a general average of 54 pct, it is evident that this method was not very accurate. A Foxboro Stabilog recording liquid density controller, Model 40, with a density range of 1.10 to 1.40 sp gr, complete with an operated Foxboro butterfly control valve, was installed to dilute the thickener slurry automatically to 35 pct solids, or 1.283 sp gr. Slurry and diluting water pipes are located at one end of the mixing tank in which this is done and a sampling pipe for the density control instrument is located at the other or discharge end. This sampling pipe is 2 in. in diam and has a minimum number of bends which must be of a large radius to prevent clogging. It was necessary to install a valve in the bottom of the sampling chamber of the density control unit to provide a self-cleaning outlet. The density control instrument is very susceptible to dirt in air. After some experimentation it was found best to use two filters in series, the first having a radial fin filter insert to remove moisture and larger airborne dirt particles, and the second containing a series of absorptive discs mounted over a central supporting tube to remove the remaining finer particles.

Another interesting accessory to the flotation system is the Clarkson reagent feeder consisting of a constant speed driven disc with a number of small capacity buckets attached. With 9/16x⅜-in. buckets used, the rate of feed can vary from 0 to 11 cu cm per min.

The handling of slurry brings up several problems. One is the constant cleaning with water of piping that has been used to transport slurry. If this is not done, a sediment is deposited each time the pipe is used, and soon the pipe will have to be replaced. When tees have one unused leg that leg is soon clogged up. It is a practice to place valves as near as possible to the point of intersection. Valves present another problem. A pinch valve has been experimented with and has been found to outlast a standard valve by a ratio above 3 to 1.

There are, necessarily, a large number of centrifugal pumps which require considerable attention. Because of the abrasiveness of the slurry, there is an excessive amount of wear on pump casings, impellers, and suction plates. After considerable metallurgical experimentation, pump parts made of Ni-Hard have been found superior to those made of other metals.

It might appear that use of the flotation process for beneficiation of low-cost cement rock as required for cement kiln feed would be complicated and uneconomical. Rather, with reasonably intelligent operators, this operation soon becomes routine. While initial cost does increase capital investment, the process has proved economically successful at the Northampton plant and other plants where this type of raw material is available.

Acknowledgments

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Some Features of Current Mining Practices At Kerr-Addison Gold Mines, Ltd.

by W. S. Row

This mine is operated at 4000 to 4500 tons daily through a single shaft, with one rock hoist and one service hoist. Latest shaft construction is concrete with wooden dividers. Economics of drifters and air-leg drills are compared, as well as costs of shrinkage and sub-level stoping. Economy of large haulage equipment and underground crushing is shown.

THE property of Kerr-Addison Gold Mines is located in the Larder Lake district of Ontario about 3 miles from the Quebec boundary on the highway between Kirkland Lake and Noranda. It is roughly 320 miles due north of Toronto and 365 miles north of Buffalo, N. Y.

The property was staked in 1906 and after several development efforts finally came into continuous production at 500 tons per day in 1937. In 1939 the daily tonnage was increased to 1200 and in 1941 to 2100 tons. At the end of the war, because of the shortage of labor, tonnage was down to 1100 tons per day, rising to 2200 in February 1947. A further plant expansion inaugurated in 1946 was completed by December 1948, bringing the capacity of the mine and mill to a production rate of 4000 tons per day. Current production, with 880 men employed, is 4400 tons per day. Changes in mining methods and equipment, required by the recently completed expansion program, have yielded interesting comparative results, some of which are described in this paper.

Hoisting of ore and waste from the mine, as well as all service facilities for handling men and supplies underground, are provided for in a single vertical five-compartment shaft. Ore and waste are hoisted in 12½-ton skips in balance by a 14-ft diam hoist with drums 115 in. wide, grooved for 2⅛-in. wire ropes. This hoist, with a total rope pull of 85,000 lb, was designed to operate at a speed of 2600 ft per min to a depth of 4000 ft, hoisting 15 tons of ore in each skip. It is amplidyne-controlled and powered with dual drive dc motors of 2250-hp each, through flexible couplings to two pinion shafts, one on each side of the main gear. Each motor has a peak load rating of 4500-hp at point of greatest duty. Direct current power to the hoist motors is supplied from a motor generator set consisting of a 5000-hp synchronous motor driving two 1750-kw dc generators.

The loading pockets used so far are at depths of 1500 and 2725 ft respectively. Consequently the present hoisting speed is 2000 ft per min and the skips capacity only 12½ tons. Now that the shaft has been deepened the new pockets at 3934 ft will be increasingly used so that the speed and the skip size will be increased up to original specifications as required.

Some difficulty in meeting the required hoisting cycle is expected because of the shape of the skips. As now in use the cross-sectional dimensions of the inside of the skip are 4 ft 6 in. x 3 ft 8 in. To hold 12½ tons, therefore, they are 14 ft 10 in. deep. Since the skips are the conventional overturning or Kimberley type, considerable hoisting time is lost in retardation in entering the headframe dump and in acceleration when leaving. It is expected that to obtain the specified tonnage from this hoist when the skip size is increased to 15 tons it may be necessary to resort to bottom dump skips. This type of skip could reduce considerably the present excessive acceleration and retardation times.

Table I. Time Analysis of Skip-Hoisting Operation, 1951

Item	Pct
Hoisting ore and waste required	73.8
Maintenance	7.9
Inspection	2.0
Service	9.3
Operating delays	7.0

Analyses of the skip-hoisting operation are made monthly, and every effort is made to reduce controllable delays. For a nine-month period in 1951 such an analysis shows that the hoist was in operation 92.5 pct of the available time, which was allotted as shown in Table I.

Overall hoisting costs, cage and skip, for the year 1951, averaged $0.168 per ton of ore milled.

The mine is serviced entirely by means of a single cage, capacity 35 men or 15,700 lb, operating in con-

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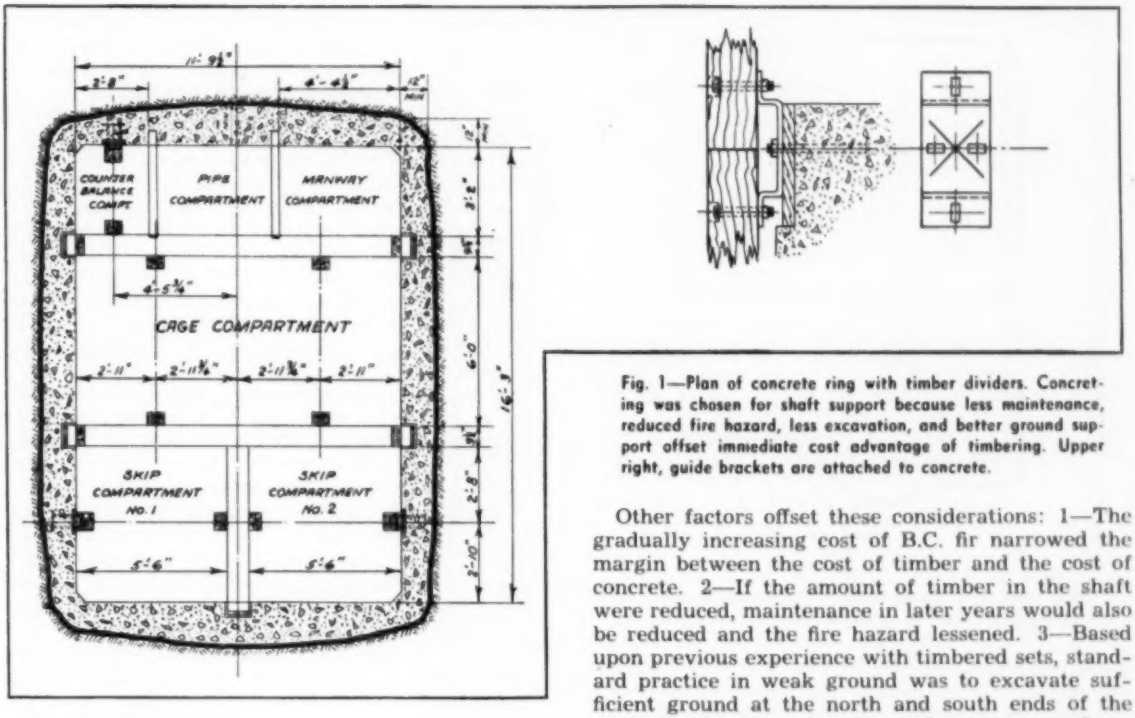

Fig. 1—Plan of concrete ring with timber dividers. Concreting was chosen for shaft support because less maintenance, reduced fire hazard, less excavation, and better ground support offset immediate cost advantage of timbering. Upper right, guide brackets are attached to concrete.

junction with a counterbalance. The cage is at present single-decked, with provision made to add a second deck below. Made of aluminum, it is 10 ft 9 in. long by 4 ft 11 in. wide, inside measurement, and operates on four wooden guides. The cage hoist is a double-drum, single-clutched, electrically driven hoist, designed to operate at 2000 ft per min to a depth of 4150 ft. The designed rope pull is 33,100 lb. The drums are 10 ft in diam by 72 in. and grooved for 1½-in. ropes. Originally the hoist was driven by an 800-hp ac motor, operating the hoist at 1280 ft per min. Recently another 400-hp motor has been added, driving through a pinion on the other side of the main gear and bringing the hoisting speed up to 2000 ft per min. The control of the two motors has been arranged for dynamic braking. To date this has been very effective and has eliminated the heavy duty previously required of the brakes, even though the hoist is now operating at a much higher speed.

The No. 3 Shaft

The shaft, dimensions of which to outside of timber are 17 ft 6 in. by 13 ft ½ in., is timbered with 10x10 in. B.C. fir sets at 6-ft intervals to a depth of 2800 ft, where sinking operations were stopped in 1942. When sinking was resumed early in 1950, consideration had been given to the relative merits of timber vs concrete as a means of providing adequate ground support. It was finally decided to use a series of concrete rings 5 ft deep at 7-ft centers, leaving a 2-ft space between rings, see Fig. 1. Timber dividers were to be cast in the top of the concrete rings, brackets to be attached to the concrete where necessary to support conveyance guides. The following points were examined: 1—The overall cost would be higher than with timbered sets. 2—Progress would be slower. 3—Since there were no company shaftmen with experience in concreting, a complete crew would have to be trained.

Other factors offset these considerations: 1—The gradually increasing cost of B.C. fir narrowed the margin between the cost of timber and the cost of concrete. 2—If the amount of timber in the shaft were reduced, maintenance in later years would also be reduced and the fire hazard lessened. 3—Based upon previous experience with timbered sets, standard practice in weak ground was to excavate sufficient ground at the north and south ends of the shaft to provide room for jacket sets to release pressure on the end plates. This would not be necessary with concrete, resulting in a saving of 9 tons of rock excavated per vertical ft of shaft. 4—Concrete provides uniform support over a wide area, not the localized support of blocking from timbered sets. This is important in soft ground, which flows around the end of the blocking. In blocky ground the more uniform support greatly reduces overbreak if the concrete is kept close to the bottom.

Plans to resume sinking were based on a shaft concreting operation at another mine. In April 1950 actual sinking was resumed, on a 2-shift per day basis, under a bulkhead at the 2800-ft level.

Since the concrete was to be carried to within 10 or 12 ft of the bottom, it was decided to adopt a cut and square drilling pattern to minimize the danger of damaging the concrete and to allow as much time as possible after the concrete was poured before blasting. Tungsten carbide-tipped rods and bore pluggers, 2 9/16 in., were used for all drilling.

Mucking was done by hand, as the various types of mechanical muckers were not readily adaptable to this particular job. The mucking performance per man was far below what had been anticipated, so in October the drilling pattern was changed to that of benching, see Fig. 2. Immediately there was a decided improvement in the breaking and mucking performances, with the crew mucking one bench and drilling and blasting another in one shift. Approximately 75 to 80 pct of the muck could be scooped into the buckets.

Initially three 21-cu ft sinking buckets were used. On Nov. 27, 1950, when the shaft had reached a depth of 3250 ft, larger buckets with a capacity slightly in excess of 1½ tons each were installed to lessen the hoisting delay, which was becoming progressively greater as the depth increased. During the benching procedure in November 1950, when muck was being scooped into the smaller buckets, the performance was 6.15 tons mucked per man

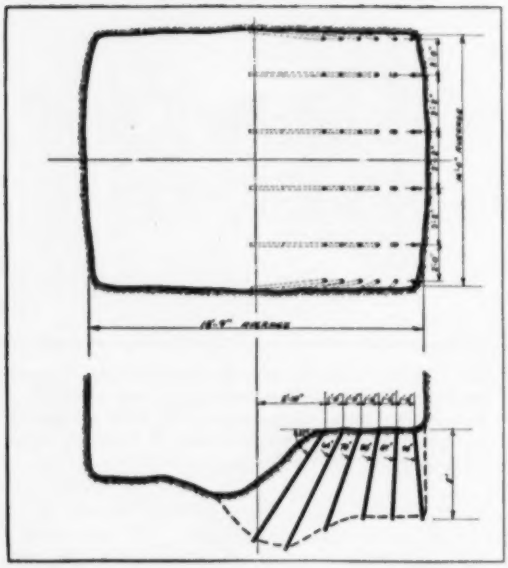

Fig. 2—Diagram of bench round which improved breaking and mucking performance in shaft.

shift. In December this increased to 6.25 tons and the average from January to September 1951 was 6.73 tons per man shift, despite the increase in depth of the shaft.

Concreting

The erection of forms and the pouring of the concrete is one phase of the operation which might cause some concern when first viewed by any operator. However, it was found that with prefabricated forms, a crew after acquiring experience could form and pour a complete ring in an 8-hr shift. Shaft sinking data will be found in Table II.

A steel supporting ring was constructed using a 10-in. wide flange beam. Both sides of the beam were filled with timber and covered with plate, the bottom side with ¼-in. plate and the upper side with ⅛-in. plate. Welded along the inside of the ring are pockets into which 2x4-in. I-beams are fitted. The latter serve as vertical ribs to hold the 2x8-in. fir form planks in place, see Fig. 3.

Four 3-ton capacity chain blocks, each with 60 ft of chain, are used to lower the ring into position. At a point 7 ft below the last concrete the steel ring is properly aligned by the use of four plumblines, hung from control points set by the engineers. Blocking is placed between the ring and the rock walls and the area between the steel and the rock is floored over. Dowel pins 12 in. long are then placed in holes adjacent to each rib pocket. These protrude 2½ in. below the concrete and are used to hold the top of the 2x4 I-beams for the succeeding concrete ring.

Horizontal ⅝-in. reinforcing rods are then placed at 9-in. vertical intervals with ½-in. vertical supporting rods at approximately 2-ft centers. In areas where considerable pressure is anticipated, rods are placed at 5-in. vertical intervals.

Form planks are then installed and five heavy hanging rods are attached to the bottom of the steel supporting ring, securing it to the set immediately above and thus relieving the chain blocks of any undue strain. At this point pouring starts, while certain members of the crew continue installing the dividers and setting the anchor bolts for the guide support brackets. The proper overlapping of these different phases of the operation is essential if the pour is to be completed in an 8-hr shift.

To carry out the pouring operation, a steel hopper is lowered from the deck and is located two sets above the ring to be poured. From this hopper is suspended a sectionalized steel pipe which conveys the concrete to any part of the forms. Concrete is mixed on the sinking deck and is lowered in a 1-cu yd bucket having a conical bottom with an arc gate through which the concrete may be discharged into the hopper, thence through the pipe into the forms.

A good quality of gravel is necessary, and since several hundred yards of gravel may be in storage at one time, control over the quality must start right at the pit. Seven bags of cement per yd of

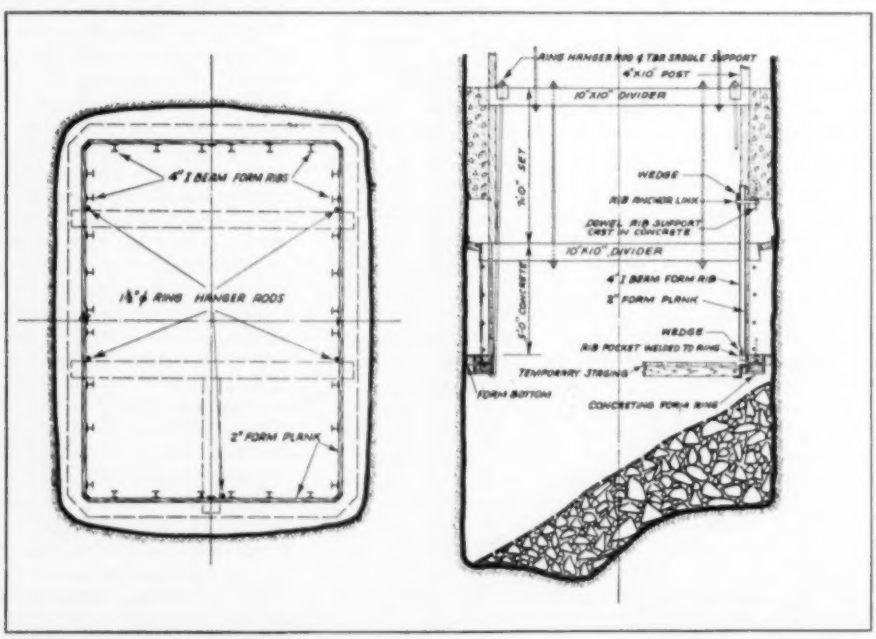

Fig. 3 — General arrangement of forms used in concreting shaft.

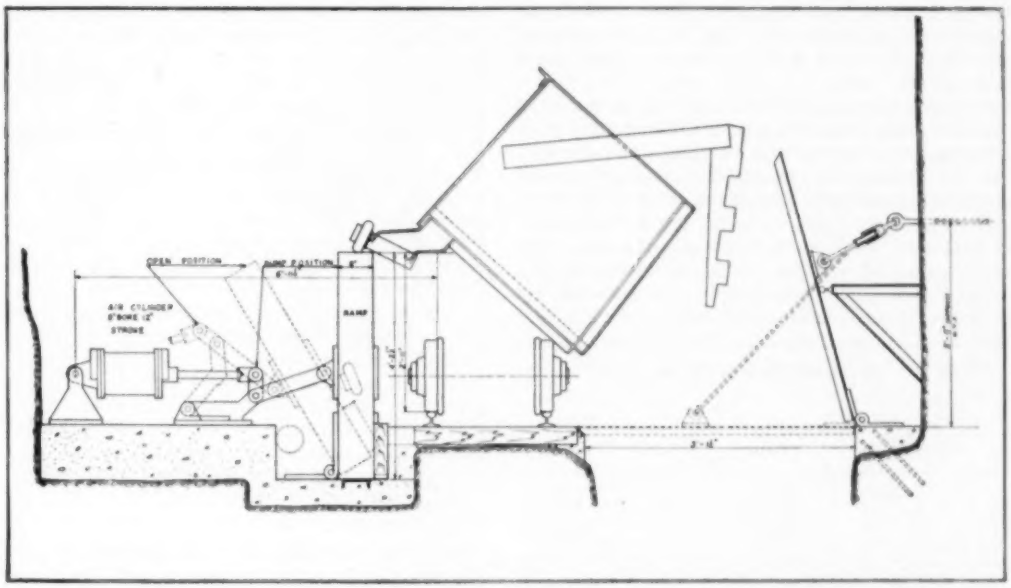

Fig. 4—Dumping arrangement used incorporates air-operated horizontal steel door to close opening, confine dust. Hinged ramp for dumping Granby-type cars can be tilted to pass cars without dumping.

concrete were used and to this mix were added ½ lb of pozzolith and 1 lb of calcium chloride per bag. Test cylinders were taken from every third ring and tests revealed a compressive strength in excess of 3500 lb per sq in. after 28 days. Excavation was sufficient to allow a minimum wall thickness of 12 in. of concrete in the ring and 18 in. in the center of the north and south walls, which are parallel to the schistosity.

Since the entire crew could not be used effectively for cutting stations, this operation was deferred until sinking was completed. Also it has been demonstrated, in this operation and others, that interruption of the sinking cycle to cut stations as each level is reached lowers the efficiency of both the sinking and the station cutting operations. In October 1951, with crews of 8 men on a 3-shift basis, work was started on station excavation. That part of the station adjacent to the shaft only is being excavated now to provide sufficient room to install the shaft station timber and concrete. Upon completion of all the stations to this stage, the skips and service cage can be operated to the 3850 level, at which time the excavation of the stations to full size will be finished and the initial development work will start.

Table II. Comparative Shaft Sinking Data

Item	Cut and Square, April-Oct. 1950	Benching, Jan.-Sept. 1951
Avg no. men per crew	10.6	11.7
Ft advance per man shift, break only	0.53	0.74
Tons mucked per man shift	4.88	6.73
Man shift per concrete ring	18.38	17.04
Total man shift per ft advance	10.53	7.40
Powder, lb per ft advance	36.6	39.6
Maximum advance per month, ft	68.0	98.0

An ore pass system continuous from the top of the mine to the bottom is located between the shaft and the ore zone. The system is composed of a series of connected 8x12 raises driven at 65° from short drifts off the main crosscuts. Below each level a finger raise from the pass leads up to the main crosscut to provide a dump for the ore trains. Over this raise is a horizontal steel door hinged on one side and opened by an air cylinder. This door covers the opening when no trains are dumping and confines the dust stirred up in the operation of the pass. At each dump also is located a hinged ramp which is locked in place while in use for dumping the Granby-type cars, see Fig. 4. When it is necessary for these cars to pass without dumping, an air cylinder tilts the ramp sufficiently to allow clearance for the cars to pass.

Ore Pass System and Crusher Station

At each level a control chute is installed in the ore pass system located in a 10x12 ft station at track elevation, see Fig. 5. Access to this station is by the aforementioned small drift, from the main crosscut, equipped with a dust-tight steel door. The throat opening of this chute is 8 ft wide by 6 ft normal to the flow of the ore. The chute is constructed entirely of concrete and is lined with 3-in. abrasion resistant steel plates 2 ft square. These plates are attached to the concrete by means of 1⅜-in. bolts into inserts cast in the concrete, see Fig. 6. Thus these plates can be easily replaced when worn out. Formerly 1-in. diam bolts were used but were found to be too small. Each 3-in. plate weighs 450 lb, and for one chute 26½ tons of steel liners are required. Flow of the broken ore through the chute is controlled by a chain curtain made up of links of 3-in. anchor chain. The total weight of the chain curtain is 5½ tons; it is operated by a 14-in. diam air cylinder with an 8-ft stroke and controlled by a 3-way valve located outside the door in the connecting drift.

At the bottom of the ore pass system just below the 2500-ft level is located the crusher station in which the bulk of underground crushing is now done, see Fig. 7. The jaw crusher, 42x54 in. in size, is fed by a vibrating type of feeder and grizzly with 6-in. openings. Ore to the feeder is controlled by a chain gate which is operated by an air cylinder. The

crusher and feeder are totally enclosed by a dust-tight metal housing connected by exhaust ducts to a dust filter. A door with an armor plate glass window allows access to take wood out of the crusher. While the rock in the vicinity of the crusher station was fairly competent, part of the station roof required support. To insure a minimum of maintenance in the future, the entire roof was supported with reinforced concrete, see Fig. 8. The walls were likewise supported with heavily reinforced horizontal concrete rings and vertical columns. The walls at each end of the station were concreted solidly and the other two walls were covered by gunite between the columns and rings.

Table III. Comparative Shrinkage Stoping Data

Year	Type of Drill	Tons Broken	Tons Per Man Shift	Powder Per Ton, Lb
1950	Drifter	739,000	45.2	0.38
1950	Air-leg	147,000	60.0	0.38
1951	Drifter	355,000	39.4	0.41
1951	Air-leg	543,000	55.8	0.36

Excavation of the crusher station was started at the bottom from the previously driven ore pass raise and mined out by shrinkage methods. Concreting of the roof and the reinforced rings and columns were carried out progressively from the top down by drawing the broken rock down as required.

Secondary breaking is reduced to a minimum, and costs thereby held down if the ore is introduced to crushers in the coarsest possible state. However,

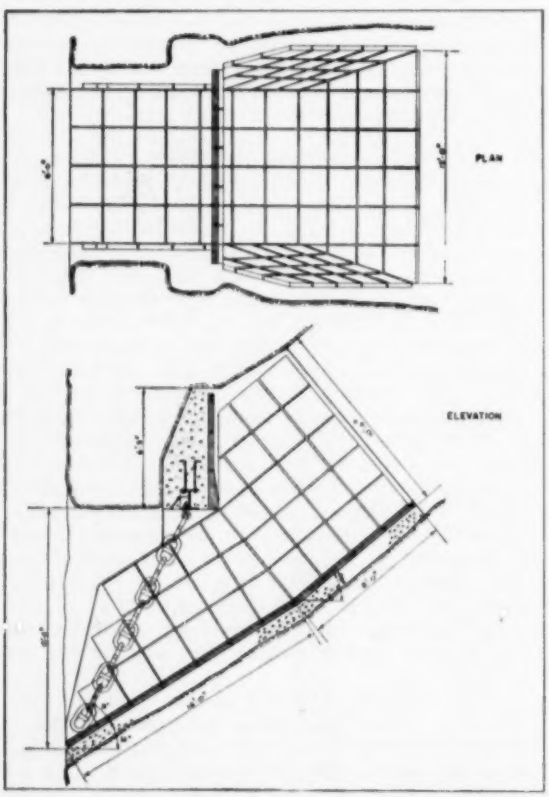

Fig. 5—Diagram of control chute installed in ore pass system at each level.

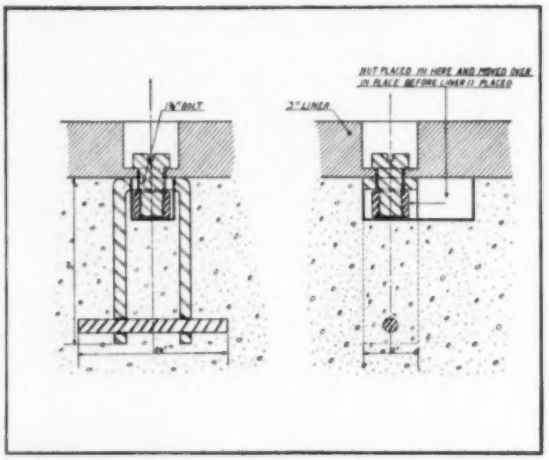

Fig. 6—Mounting for abrasion-resistant steel inserts in a concrete chute.

very coarse ore does not flow freely through control chutes and measuring pockets and into skips. The handling of large chunks of ore in skips results in lowered hoisting efficiency and higher maintenance costs of both skips and shaft as compared to an operation where crushed ore is hoisted. For these reasons primary crushing has always been done underground at Kerr-Addison.

Stoping methods in use during 1951, see Table III, accounted for the following percentages of ore broken during the year: shrinkage 77 pct, sub level 16 pct, cut and fill 2 pct, and pillar recovery 5 pct.

Shrinkage Stoping

Shrinkage stoping has accounted for the greater percentage of mill feed to date. Wide orebodies on the upper levels have been mined by a series of transverse stopes, but general practice is to lay out the stopes with their greatest dimension along the strike of the orebody. Good ground has made it possible for widths up to 150 ft to be mined by shrinkage stoping operations. Normally orebodies under 30 ft in width are developed at 150-ft vertical intervals; those wider than 30 ft are mined through a 300-ft vertical lift, see Fig. 9.

A regular pillar system provides temporary support to the walls during mining operations. These pillars are mined by long-hole drilling when the stopes on each side are pulled empty or by cut-and-fill methods under certain conditions.

The year 1951 has seen all drifters replaced in stopes with air-leg drills using tungsten carbide-tipped rods, the bits of which are chisel-shaped. As was anticipated, the changeover resulted in an appreciable increase in tons broken per man shift.

The lighter equipment has materially reduced the frequency of accidents caused by strains through carrying heavy equipment over relatively rough muck surfaces in stopes.

Data over a period of nearly 2 years, see below,

Period	Frequency Per 100,000 Shifts		Severity Per 100,000 Shifts		Rating	
1950 & 1951	Drifter	Air-Leg	Drifter	Air-Leg	Drifter	Air-Leg
	1.45	0.42	47.7	10.0	69.0	4.2

shows that not only is the accident frequency lower with air-leg drills but that the accidents which do happen are less severe.

Short period delay blasting caps are now used in the majority of shrinkage stopes. This appears to have resulted in better fragmentation and in less lost time from cutoff holes, but as yet insufficient data are available to indicate any definite decrease in secondary breaking costs.

Sub-level Mining

Sub-level mining methods have been applied chiefly to flow-type orebodies where the ore outline is more regular than in the carbonate zones and where walls are sufficiently strong to stand over a long period of time. The regular pillar system used with shrinkage mining lends itself to sub-level mining as well. All drilling is done from hanging wall and foot wall drifts, holes being laid out in a radial pattern with rings at 7-ft intervals, see Fig. 10. Within the radial pattern, tonnage per foot of hole drilled equals 3.0, but because of the slightly smaller burden in the slot area and also because of added holes on the walls, the overall tons per foot of hole averages 2.54. All blasting is done with 75 pct, 1¼x8 in. Dygel. Short period delay blasting caps are used. Average powder consumption is 0.25 lb per ton.

Initial sub-level mining was done with diamond drills, working from sub-levels at 75-ft intervals. Sectional steel rods with 1¾-in. tungsten carbide bits were tried under similar conditions, but company experience had proved that 50 ft was the economic limit to which a hole could be drilled. Consequently the sub-level interval was reduced to 50 ft with certain definite advantages: 1—No holes longer than could be drilled economically with sectional steel were necessary. 2—The shorter interval permitted closer geological control, resulting in a more complete extraction of ore where irregularities occurred in the ore outline. 3—Slot raises could be driven from sub level to sub level without timber. 4—The proximity of the first sub level to the box-holes permitted the sill cut to be mined by long hole drilling. 5—While the maximum distance is maintained between the toe of adjacent holes in any one ring, the efficiency of the collar of the holes is increased slightly as the sub-level interval is decreased. 6—The 50-ft interval fitted in with the 150-ft main level interval, placing every third sub level at a main level.

Table IV. Comparative Sub-Level Stope Drilling Data

Jan.-Nov. 1951	Footage Drilled	Labor, $	Rods, Bits, Drill Mtce., Comp. Air, $	General Expense, $	Total Cost Per Ft, $
Diamond drill	97,516	0.29	0.46	0.18	0.93
Sectional steel	20,855	0.26	0.23	0.17	0.66

Diamond drills are now being replaced in sub-level stopes by 3½-in. drifters with automatic feed, 1⅛-in. hexagonal sectional drill rods in 30 and 60-in. lengths, and 1¾-in. tungsten carbide bits. Couplings are 3 in. long and 1⅝ in. OD, with reverse buttress thread.

A complete conversion to sectional steel will not be possible, since to obtain geological information from two to three holes are diamond drilled on every third ring. This may account for as much as 10 pct of the entire footage drilled.

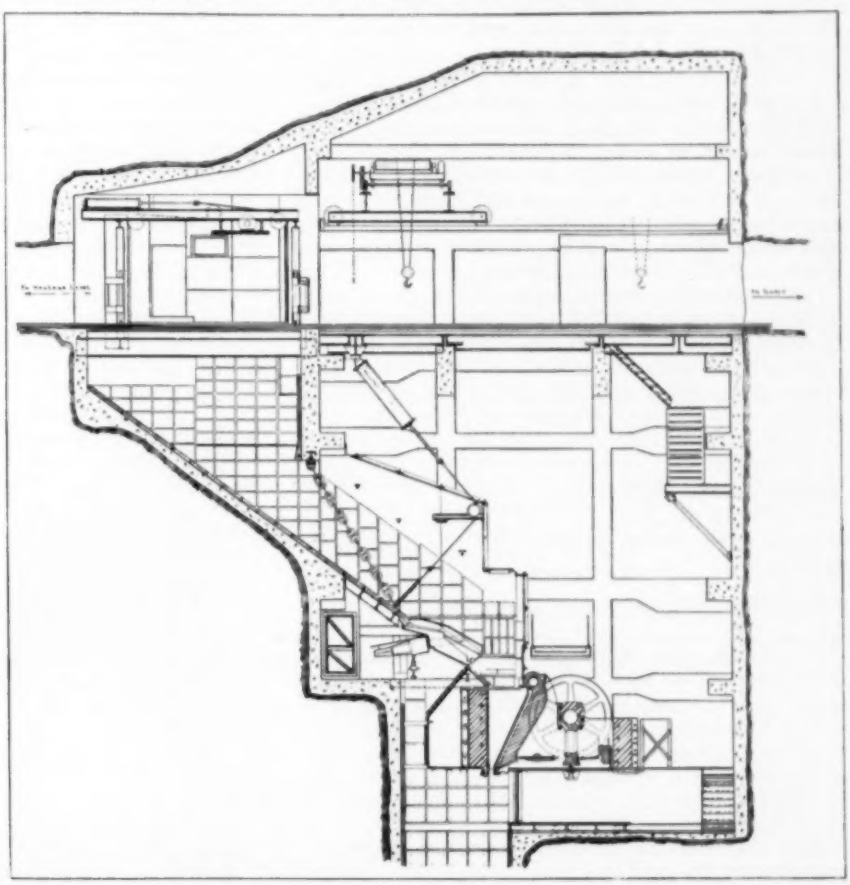

Fig. 7—Underground crushing station at 2500-ft level at bottom of ore pass system. The 42x54-in. jaw crusher is fed by vibrating feeder.

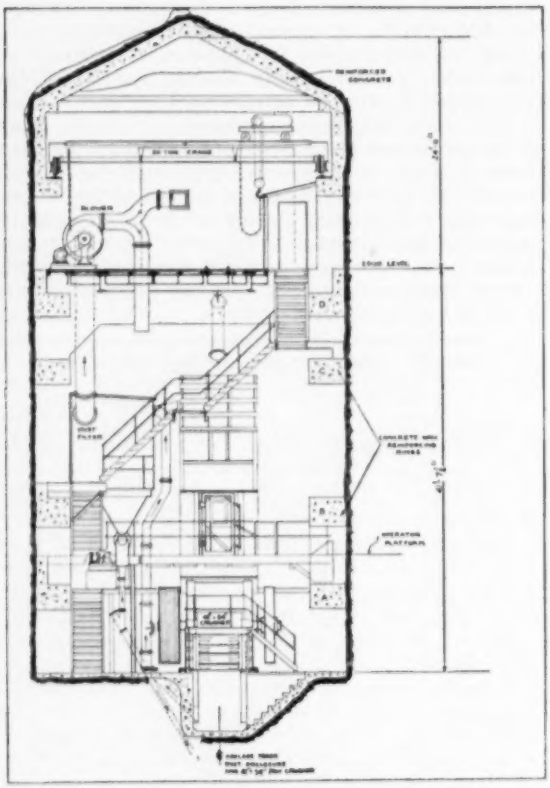

Fig. 8—Roof of underground crushing station at 2500-ft level is supported by reinforced concrete. Walls have heavy concrete rings and columns.

Table IV offers a comparison of sub-level stoping drilling costs by diamond drill and sectional steel from January to November, 1951.

Of interest is a comparison of the cost of mining a block of ore by 1—shrinkage mining, with air-leg drills and tungsten carbide bits and drifters using steel detachable bits, and 2—sub-level mining, with diamond drill and sectional steel.

To obtain a fair comparison, a block of ore 50 ft wide, 100 ft long, and 300 ft high was assumed, and to this the above mining methods were applied,

Table V. Cost Comparison for Mining Block of Ore by Various Methods

Mining Method	Stope Dev., $	Mining, $	Total	Total Tons	Cost Per Ton, $
Shrinkage, air-leg	9,561.00	68,630.72	78,191.72	106,190	0.74
Shrinkage, drifter	9,561.00	78,131.52	87,692.52	106,190	0.83
Sub-level stope, diamond drill	31,243.37	55,454.88	86,698.25	105,712	0.82
Sub-level stope, sectional steel	31,243.37	45,699.20	76,942.57	105,712	0.73

using costs based on actual operations from January to November, 1951. Table V gives a summary including all stope development, with the exception of the scram drift, and all mining costs. It is assumed in each case that all ore is handled from the stope to the transfer raise through a scram drift.

It is felt that as the miners become more accustomed to the air-leg drills a greater reduction in shrinkage costs can be effected. Sectional steel has been in use since June, 1951, and accounts for a footage approximately one fifth of that drilled by diamond drills. Each month has seen a reduction in cost per ft of hole drilled with sectional steel, so there is justification in predicting further reduction of sub-level mining costs by its use. The slightly larger hole resulting from the sectional steel drilling as compared to diamond drilling permits a heavier powder concentration per ft of hole; consequently in current drilling practice the interval betweeen rings is being increased from 7 to 7½ ft, a move which should reduce further the overall cost per ton.

Cut-and-Fill Mining

Horizontal cut-and-fill stoping as yet accounts for only a small percentage of the total tonnage mined or broken. This method is confined at present to one orebody in the hanging wall between the 300-ft level and surface. Eight-foot cuts are taken with stopers, holes being drilled 10½ ft deep at 50° and blasted with short period delay blasting caps. Better fragmentation and drilling cycle through this practice, see Table VI, has improved the overall performance above that obtained with horizontal holes and standard delay caps or safety fuse.

Table VI. Comparative Performance Data in Cut-and-Fill Stopes

Method of Drilling	Tons Broken Per Man Shift	Powder Per Ton, Lb	Tons Slushed Per Man Shift	Powder Per Ton, Lb, Sec. Break
Flat holes, drifter	32.8	0.46	29.6	0.07
Inclined holes, stoper	35.6	0.50	65.8	0.03

Ore is scraped from the stopes and the fill is spread with a double-drum 15-hp air hoist and a 48-in. scraper. Chutes are built of 8x8-in. cribbing and lined with round lagging flattened on one side. Round lagging at 10 to 12 in. centers is used for flooring and waste rock from the mine is used for fill. It is anticipated that cut-and-fill mining will be used to a much greater extent when greater depths are reached. To this end considerable re-

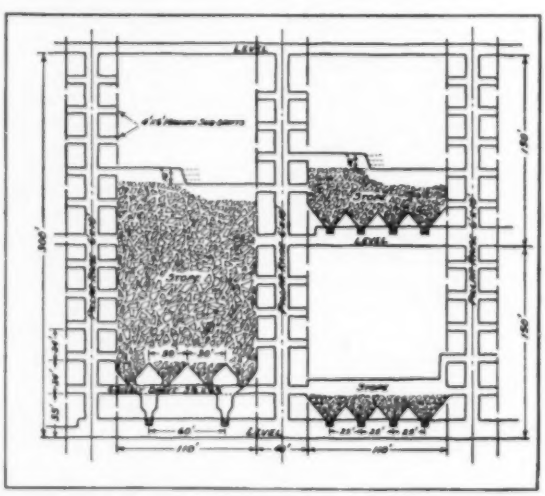

Fig. 9—High sill shrinkage stope, left, and low sill shrinkage stope, right. Orebodies under 30-ft width are developed at 150-ft vertical intervals, wider orebodies at 300-ft intervals.

search has been directed toward the classification of mill tailings to produce a satisfactory fill. Initial work has been encouraging.

Stope Pulling Layouts

To provide facilities for handling broken ore from the stopes into the ore trains, three types of stope development are in use at Kerr-Addison: 1—low sill development in which the ore from stopes is drawn directly into the ore trains, 2—high sill development in which grizzly chambers under the stopes are used to condition the muck before it is loaded into the ore trains, and 3—scram drifts under stopes in which the ore is slushed to a transfer raise and loaded into the ore trains.

Low Sill Development: Low sill development is used in narrow shrinkage stopes or shrinkage stopes that will be 150 ft or less in height. Following the drift development through the orebody, the drift backs under a stope are taken down to a height of 14 ft 6 in. above the rail. Boxholes are then driven up to the sill elevation of the stope 25 ft above the track and are located so that the distance between the draw points in the stope will not exceed 25 ft. The boxhole is then coned out until connected with adjacent boxholes. After the second round in the boxhole is blasted, a standard timbered chute, partly prefabricated on surface, is installed, see Fig. 11. This chute is 4 ft wide in the clear and is supported on a platform which covers the whole drift and is located to provide 7-ft clearance above the rail. Muck flow through the chutes is controlled by a bumper log and a hand-operated arc gate. To reduce chute maintenance, blasting of large pieces is done in the rock throat of the chute as much as possible. The trammers pulling the chutes work entirely from the platform. From a cost point of view, performance in this type of operation is good. This is due in part to the fact that there is better fragmentation in smaller stopes. However, haulage equipment is not used so advantageously as in other methods and thus production rate is limited.

High Sill Development: High sill development is used under large shrinkage stopes where fragmentation may be coarse and large tonnages will be handled. Grizzly chambers are located at an elevation of 35 ft above the haulage drift tracks. Orientation of the grizzlies is such as to provide the most favorable location of the draw points at 30-ft centers, with respect to the ore outline. Access to the grizzly chamber is through a subdrift connected to a service raise in the adjacent pillar.

Each grizzly chamber is 16 ft long and is equipped with a double-end grizzly made of 12x14-in. B.C. fir covered with 12-in. steel channels. On each end there are two openings 20 in. wide by 6½ ft long. Under the grizzly a raise slashed out to hold at least 20 tons connects with a standard boxhole chute in a haulage drift as described under low sill haulage. From the grizzly 5x7-ft finger raises are driven to the stope sill elevation 20 ft above the grizzly level, where they cone out to 30 ft and interconnect.

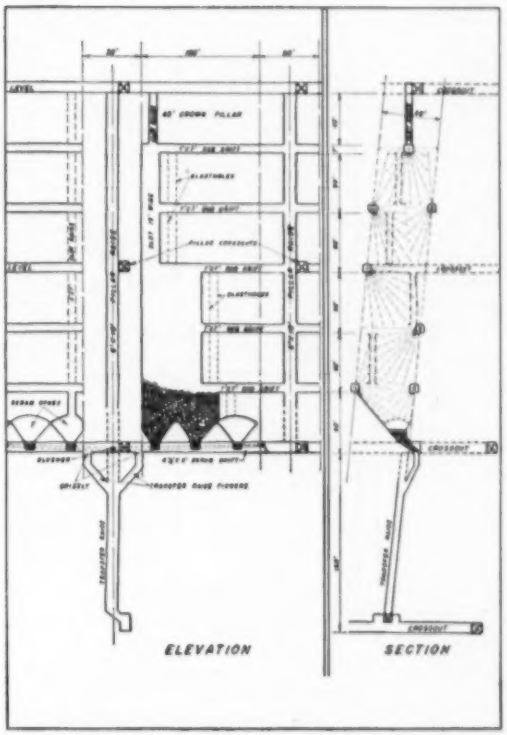

Fig. 10—Drilling for sub-level stoping is done from holes in radial ring pattern at 7-ft intervals from hanging wall and footwall drifts.

The main advantages of high sill over low sill haulage are that a much greater rate of production is possible, secondary breaking of the coarse ore from large stopes is facilitated, and the operation is safer than low sill haulage.

Scram Drifts: Scram drifts with scrapers handling the ore into transfer raises are used exclusively under sub-level stopes and under some of the more recently developed shrinkage stopes. General practice now is to drive the scram drifts at level elevation parallel to the strike of the orebody and along its foot wall. Initial practice was to locate them in the center of the orebody, but maintenance costs were high owing to excessive spalling of the scram drift backs and brows in the boxhole after stoping operations commenced. Originally the scram drifts were driven 8 ft wide by 9 ft high, but this width was found to be too great for a 66-in. scraper, and now the width has been reduced to 6½ ft.

From the level below that on which the scram drifts are located a transfer raise is driven on the center line of the pillar separating adjacent stopes.

Table VII. Comparative Data on Handling Broken Ore in Three Types of Stope Pulling Layouts

	Low Sill		High Sill				Scram Drift			
	Total, $	Tons	Grizzly, $	Haulage, $	Total, $	Tons	Scram,$	Haulage, $	Total, $	Tons
Labor	0.14		0.12	0.07	0.19		0.07	0.05	0.12	
Total	0.33		0.28	0.15	0.43		0.33	0.10	0.43	
Production per man shift		72				50				83
Production per tr. crew shift		216				429				582

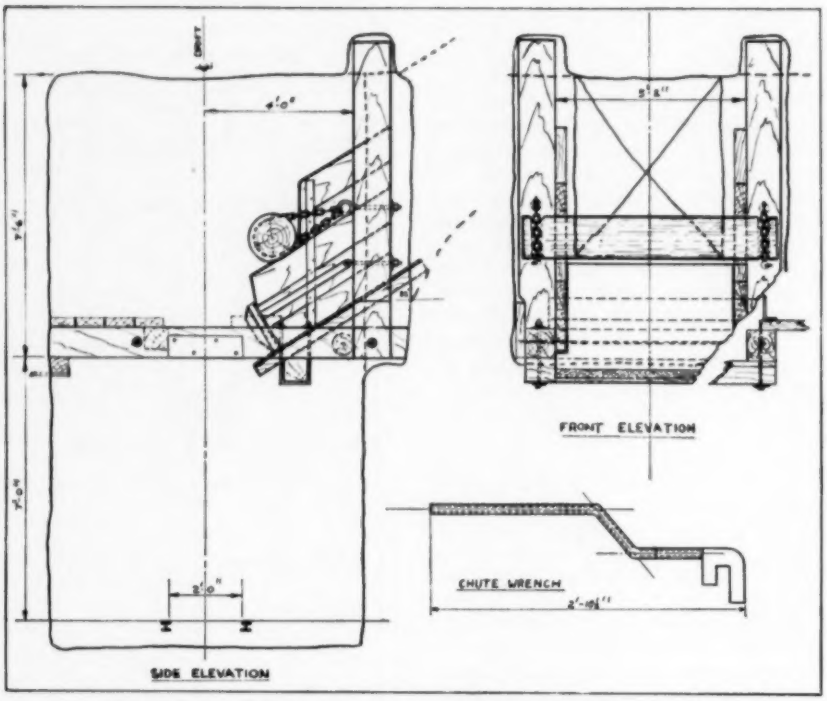

Fig. 11—Standard timbered boxhole chute has sides and bottom prefabricated in shop.

The collar of this raise on the haulage level below is equipped with an efficient steel and concrete chute with an air-operated undercut arc gate, see Fig. 12. The chute has a throat opening 5 ft wide by 4 ft normal to the flow of the ore. At the top of this raise finger raises are driven to serve as many as four drawpoints in adjacent scram drifts. The tops of the finger raises are protected by a discarded cone crusher liner having an opening 3½ ft in diam, set in concrete. General practice is to locate the scram drifts under adjacent stopes along the strike of the orebody so that one permanently located slusher hoist will handle the broken ore from two stopes by simply reversing the direction of the cables.

Table VIII. Comparative Efficiency and Direct Labor Cost Of Ore Haulage in Different Types and Sizes of Cars

Haulage	Tons Per Man Shift		Direct Labor Cost Per Ton	
	90 Cu Ft	110 Cu Ft	90 Cu Ft, $	110 Cu Ft, $
Low sill	50	57	0.20	0.18
High sill	97	131	0.105	0.075

From the scram drifts, boxholes 6 ft wide and 4½ ft high are cut at approximately 30-ft centers and driven at 45° to the sill elevation. They are then coned out and the center of the brow is opened up a bit to form an arch. Two-drum, 60-hp electric hoists having a rope pull of 8150 lb at 240 ft per min are used for slushing. Discarded hoisting ropes ⅞ to 1 in. in diam are used and wire rope cost, per ton scrammed, averages 6 to 10¢. The scrapers used are 66-in. hoe type and weigh 2735 lb. As stated above, the slushers under sub-level stopes are located permanently on a base, but under shrinkage stopes slusher hoists mounted on trucks are used. This permits maximum use of equipment, since when the drawing cycle is complete in one shrinkage stope the slusher can be moved to another ready for pulling. Slusher performance under shrinkage stopes averages 297 tons per scraper shift and under sub-level stopes 274 tons per scraper shift. Advantages of the use of scram drifts are relatively high rate of production, less pillar requirement under a stope, and higher production per man shift.

In Table VII the three methods of handling broken ore from the stope to the ore pass are compared on a cost per ton and production basis.

Included in the above scramming costs are maintenance costs of 2¢ for scrapers and 8¢ for concreting scram drifts. This latter figure is expected to be reduced by the present practice of locating the scram drift in the foot wall of the ore zone.

From the above table it is evident that scramming production per man shift is higher, a very important item in the present tight labor market. Rate of production from a stope with scramming facilities is 36 pct higher than in the high sill method and 169 pct higher than from a low sill stope.

Haulage Equipment

Level haulage underground is done entirely by battery locomotives. On development work 2-ton motors are in general use and for production, 4-ton and 6-ton locomotives. The 4-ton locomotive complete with battery weighs 12,000 lb and the 6-ton complete with battery weighs 20,000 lb.

Use of battery locomotives exclusively for underground haulage was decided for the following reasons: 1—They allow complete flexibility of movement of equipment on any level. 2—Although tonnage from any level is high, it originates from several sources on a level, and thus equipment to handle it is always available. 3—Ore passes are well

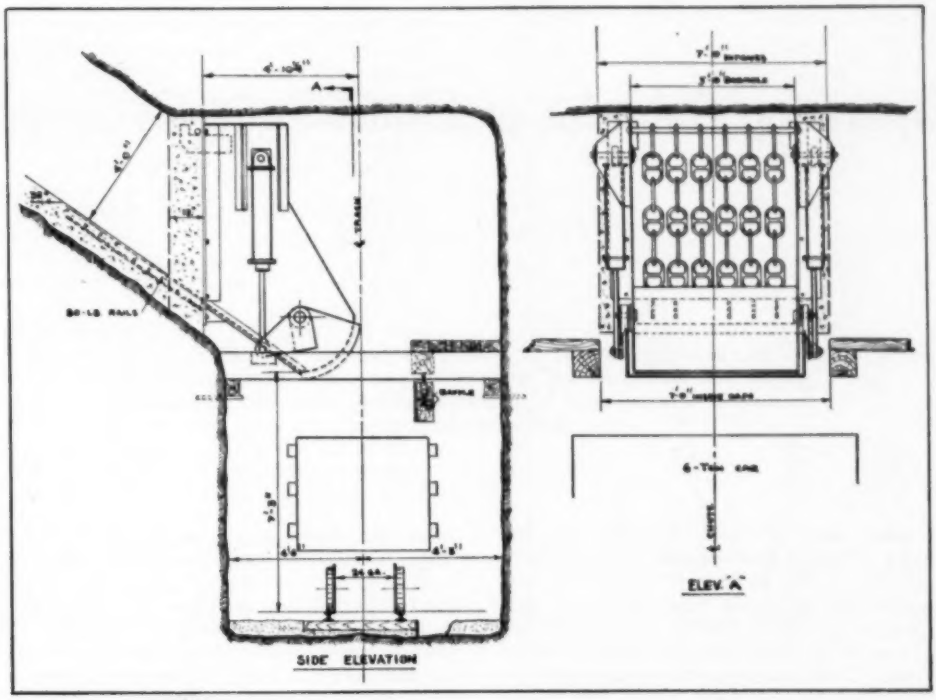

Fig. 12—Air-operated undercut arc gate is used on standard loading chute No. 3.

located so that average tramming distances are relatively short. In fact, for the first ten months of 1951 distance trammed averaged only 1260 ft. 4—There is relative safety because of low voltages used in battery as compared to trolley locomotives. 5—There is low fire hazard, as all electrical equipment is localized in level charging station with battery locomotives.

Cars used for development work are 2-ton Hudson-type and 65-cu ft Granby-type cars. For ore haulage the original cars used were 90-cu ft Bell-type design. The bottom of this car slopes at 40° toward a side opening door. The door is locked in the closed position and is released and opened at the ore pass by a wheel passing over a ramp. The car chassis is mounted on two 4-wheel trucks, a feature allowing the car to be turned at any switch. This type of car was chosen in anticipation of a 1000-ton operation because the sloped bottom would allow cleaner dumping of the ore, the fines of which were sticky. However, in more recent years, in the handling of coarser ore, much time has been lost at the ore pass because it has frequently been necessary to bar large pieces out of the cars. Moreover, the design of the car renders it topheavy when loaded. Also the plan area of the car is greatest at the top, where of course the effect of variation in loading is at its maximum.

Table IX. Comparative Performance of 4 and 6 Ton Locomotives in Three Types of Haulage

	Tons Per Man Shift		Direct Labor Cost Per Ton, $	
Haulage From:	4-Ton Loco.	6-Ton Loco.	4-Ton Loco.	6-Ton Loco.
Low sill	57	66	$0.18	$0.155
High sill	131	185	0.075	0.055
Transfer raises	183	232	0.065	0.045

For these reasons 110-cu ft Granby-type cars were purchased. The size of the service cage governed the size of the car, and the ones purchased can be loaded on the cage after the ramp wheel and couplers are removed. The car is 11 ft 12 in. long overall, 4 ft 6 in. wide, and weighs 10,500 lb. The door is hinged to swing freely to allow dumping the large pieces of ore without barring. The undercarriage of the car is cast steel and adjacent to each wheel are located a leaf spring and two helical springs. The latter are located directly over the rail and, since these cars are operated on 24-in. gage track, have an essential stabilizing effect on the car. As expected, the car factor for this car was 24 pct better than for the Bell-type 90-cu ft car.

As the 4-ton locomotives handle only five cars of either the 90-cu ft Bell or 110-cu ft Granby type, comparative performance figures for each kind of train on low and high sill haulage are available. Table VIII gives this information for the first nine months of 1951.

Thus in low sill haulage tons per man shift increase 14 pct and direct labor costs decrease 10 pct. In high sill haulage where tram equipment is used to better advantage an increase of 35 pct in tons per man shift and a decrease of 28.6 pct in direct labor costs were brought about.

By the purchase and use of the 6-ton locomotives further economies were effected, since fifteen 110-cu ft cars could be drawn in a train as compared to five drawn by the 4-ton locomotives. Table IX shows the comparative performance of the 4 and 6-ton locomotives.

Use of the larger locomotives resulted in increases in tons per man shift in low sill haulage 16 pct, high sill haulage 41.5 pct, and from transfer raises 51.7 pct. Similarly, direct labor costs per ton hauled were decreased 13.9 pct, 26.7 pct, and 31 pct respectively for the three types of haulage.

Operating Behavior of Liquid-Solid Cyclones

by E. B. Fitch and E. C. Johnson

The operating behavior of liquid-solid cyclones is outlined, together with the nature and range of the process results obtainable, to serve as a background for engineers wishing to consider application of this new process tool.

BY now most engineers are familiar with the liquid-solid or Dutch State Mines cyclone. However, it should be helpful to know exactly what it is that the equipment does and what its limits are. Without going into cyclone theory, this paper will describe the operating characteristics of Dutch State Mines cyclones. These are manufactured under license in this country and sold under the trademarked name of DorrClone. The physical construction of the liquid-solid cyclone has been covered in many papers,[1-4] the DorrClone in particular being described in some detail by Weems.[5]

Fig. 1 shows the unit in cross-section. The feed enters at C. The coarse, heavy particles are thrown centrifugally to the periphery and make their way down the wall to the apex where their rate of discharge as underflow is controlled by an adjustable rubber apex valve. As the apex diameter is decreased the solids build up behind the valve, producing a denser underflow. Meanwhile the fine particles are swept into the upward flowing vortex stream which exits as overflow through the vortex finder, F.

Flexibility to produce the specific result desired in a particular process is achieved by providing means for varying the areas of the entrance, vortex discharge, and apex discharge. The entrance area may be varied by insertion of special shims. Vortex discharge area may be changed by use of different-sized vortex finders which are interchangeable. Similarly, the different sizes of apex valves are interchangeable and in addition each apex valve is variable down to about 60 pct of its maximum diameter.

A most significant primary distinction to make is that although liquid-solid cyclones have been sometimes called thickeners, they actually are classifiers, and very potent ones. They are almost never thickeners in the special sense that many metallurgical engineers understand the term. There would be no profit in quibbling over the definition of a word, but when the application of cyclones is considered, it will help to understand the difference between two mechanisms, one of which will be called *classification*, and the other *thickening*.

In what is called thickening the fine solid particles present in the feed hold together by surface attraction during the sedimentation process. The loose network of particles thus held together constrains all particles to settle at approximately the same rate, the larger ones dragging the smaller ones down. As a result, pulp settles with a sharp line of demarcation between solids and a relatively clear supernatant liquid. Essentially all the solids, regardless of their fineness, pass into the thickened underflow, and a clear overflow is separated.

In classification, on the other hand, the interparticle forces are relatively insignificant as compared to the settling force on the individual particles, and are insufficient to prevent independent movement of the particles. The coarsest, heaviest particles settle most rapidly through the pulp, passing more slowly settling fines. Particles coarser than the mesh-of-separation essentially all settle into the underflow, but if the feed contains any particles finer than the mesh of separation, at least part of them will appear in the overflow. A clear supernatant or overflow can be obtained only if there are no undersize particles present in the feed.

Thus it will be seen that classification is impossible under ideal thickening conditions. The finer particles are pulled down at essentially the same rate as coarser particles, and there is no separation on the basis of particle size.

The surface attraction holding the particles together in a thickener is usually feeble. Whenever the sedimentation force on any particle is strong with respect to the interparticle forces, that particle can pass through the tenuous structure and settle independently. There are at least four ways of making the sedimentation force strong, with respect to the interparticle forces, and obtaining classification. First, and most obvious, the particles may be large and heavy. Thus coarse sands settle out in a beaker or Dorr thickener ahead of the rest of the thickening solids. Second, the interparticle forces may be altered by physicochemical means; i.e., it is often necessary to add dispersing agents to destroy the interparticle forces and permit classification to take place. Third, the interparticle forces may be reduced by dilution of pulp. It is well known that to obtain the most efficient separation of

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fines and oversize in a gravity-sedimentation or Dorr-type classifier it is necessary to dilute the feed to some appropriate value. Fourth, the sedimentation force may be increased through substitution of a centrifugal field of many G's for the simple force of gravity. This happens in a cyclone, and partly explains why a cyclone nearly always operates as a classifier and not as a thickener.

There is another and perhaps even more important mechanism which acts to make the cyclone a potent classifier rather than a thickener. As has been described by several writers,[1,3,4] the fluid in a cyclone does not move at constant angular velocity as it tends to do in a centrifuge, but speeds up as it moves towards the axis. The velocity gradient sets up shear forces in the fluid which overcome the natural flocculence of any but most the strongly flocculated pulps. Fine particles are prevented from attaching to larger particles and settle separately.

From the above, it will be seen that a cyclone essentially always acts as a classifier. It classifies small particles, and it can do so without the expedients of added dilution or dispersing agents. This characteristic is one of the most valuable special assets of the cyclone.

Mesh of Separation

The first question to be asked about a classifier is what separation it makes. Here there are two difficulties. To begin with, there is some confusion in terminology. In literature on cyclones the separation has frequently been designated in terms of the size particle which reports half to the underflow and half to the overflow. In metallurgical practice it is perhaps more common to designate separation in terms of the largest sized particles that appear in the overflow in any significant quantity, this being commonly termed the *mesh of separation*. In classification by ordinary gravity sedimentation, it makes little difference which point is described, as long as it is known which one is intended. In this case the 50 pct point is generally a calculable amount, in the range of the second screen partial, finer than the mesh of separation. In cyclones, as will be shown later, there is no such predictable relationship. This paper, therefore, will refer always to the mesh of separation, according to common metallurgical practice, rather than to the 50 pct point referred to in the literature on cyclones.

Table I. Separations Obtained with Various Fittings in a 3-in. DorrClone Separator Operating on a Dilute Suspension of −65 Mesh Dolomite Solids in Water, Fed into the Cyclone at 20 Psi

Feed Entrance, Area in Sq In.	Vortex Finder, Diam, In.	Separation, Microns
0.75	1.05	34
0.53	0.82	28
0.30	0.62	24
0.19	0.49	23
0.10	0.36	27

In the second place, there are a surprisingly large number of variables significantly affecting the separation made by this mechanically simple piece of apparatus. It could be misleading to give the range of separation to be expected from the different sizes of units without explaining how this may vary, and without stating fully the conditions for which the specified range would be valid. For this reason the effect of the more important variables on the mesh of separation will be discussed, and data will be given on the range of separation of the different sizes of DorrClone units under specified conditions.

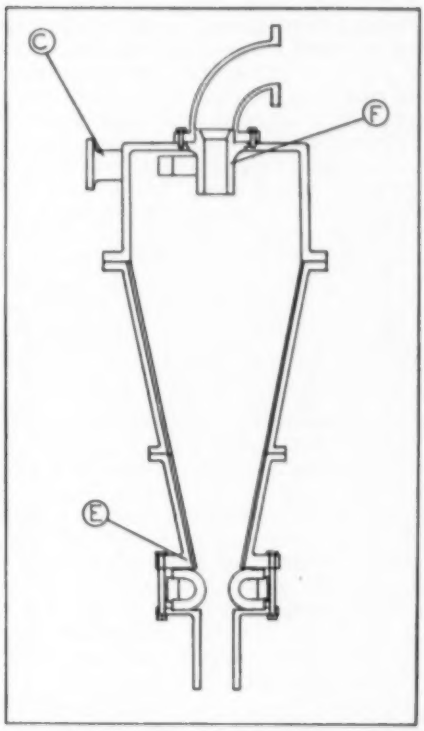

Fig. 1—DorrClone with tire-type apex.

Given a series of cyclones geometrically similar except for the sizes of the feed, and the overflow and underflow orifices, the following have been found significant variables in determining the mesh of separation produced: 1—Size of feed entrance and vortex finder, 2—Size of cyclone, 3—Specific gravity of feed solids, 4—Quantity and size distribution of feed solids, 5—Pressure drop or head loss across cyclone, 6—Plasticity of feed pulp.

Size of Feed Entrance and Vortex Finder

The prime adjustment for mesh of separation is the size of the feed entrance and vortex finder. Larger fittings give coarser separations. Smaller fittings, down to a certain critical size, give finer separations. However, for any given size of cyclone, making fittings any smaller than this critical size results again in coarser separations, rather than in finer ones. For minimum mesh of separation in any given size of cyclone, the feed entrance and the vortex finder usually have equivalent diameters in the range of 1/5 to 1/6 the diam of the cyclone. However, this has been observed to vary with the nature of the feed slurry.

To illustrate the effect of changing the feed entrance and vortex finder, Table I shows the separations obtained in a 3-in. cyclone operating on a dilute slurry of −65 mesh dolomite solids. The operating pressure was constant at 20 psi. In every case the feed entrance has an area essentially equal to that of the vortex finder. Each point is the average value for seven to nine individual test runs.

Smaller cyclones can make finer separations than larger ones; they cannot be fitted to give separations as coarse. This will be apparent from the data on the range of separations possible with the different

sizes of cyclones. Thus the minimum separation obtained with a 12-in. cyclone under the conditions as specified for Table I was 37 microns, as compared to the minimum of 23 microns shown in Table I for the 3-in. unit.

Table II. The Effect of Dilution on Separation

Dilution in Cyclone Overflow, Volume	Solids in Cyclone Overflow, Wt Pct	Separation
4	42	coarser than 147 mu, 100 mesh
5	37	60 micron
6	32	43 micron, 325 mesh
10	22	36 micron
20	13	32 micron
Very high	Low	29 micron

It is obvious that the specific gravity of the solids affects the separation size. Dahlstrom's correlation[2] showing separation inversely proportional to $(\rho_s-\rho)^{1/2}$ has proved satisfactory where ρ_s = sp gr solid and ρ = sp gr slurry.

The mesh of separation obtained in a cyclone is strongly affected not only by the quantity of solids in the slurry treated, but also by the size distribution of the solids. As the solids become crowded in a slurry the particles interfere with one another and settle more slowly. The mesh of separation becomes coarser. Increasing coarseness of the solids also coarsens the mesh of separation.

The effect of dilution is illustrated qualitatively in Table II. In this case the feed was a 95 pct −325 mesh dolomite slurry of varying dilution. A 12-in. cyclone was used. Separation is here correlated with overflow dilution, although it is not established that this is the best correlation to use. In Table II volume dilution is defined as the volume of liquid in a slurry divided by the volume of solids.

While it is not safe to draw any quantitative conclusion from Table II, it is clear that as solids become crowded in a slurry, separation becomes coarser.

Experience shows that the effect of dilution can usually be disregarded for purely practical purposes until the volume dilution of the feed drops below 8. At some dilution less than this it begins to be very important. At just what dilution the rapid increase in diameter-of-separation sets in seems to depend upon the size of the unit, the fittings used, and probably upon the coarseness of the solids. Table III

Table III. Percent Solids at Volume Dilution of 8

Solids, Sp Gr	Solids, Pct
1.3	14
2.0	20
2.7	25
3.0	27
4.0	33
5.0	39

shows the weight percent solids at a volume dilution of 8 for solids of different specific gravities. If the percent of solids in a slurry is much higher than shown in Table III, separations substantially coarser than those to be mentioned below must be anticipated.

The effect of coarseness of feed can be seen from the following examples. Under essentially identical operating conditions, a 3-in. DorrClone unit gave a 23-micron separation when treating −65 mesh dolomite solids, and a 16-micron separation on −325 mesh solids. A 12-in. DorrClone unit gave a 37-micron separation on −65 mesh dolomite solids, and a 29-micron separation on −325 mesh solids. Even though the difference in feed did not appear to be great, there was a substantial difference in the separation. It will be apparent, therefore, that separations in a cyclone may be made substantially finer if the feed is first scalped of coarse particles. The scalping device may be another cyclone.

Increasing pressure on a given cyclone results in finer separations. As a generality the diameter-of-separation seems to vary inversely as something less than the fourth root of pressure; thus doubling the pressure would give as a maximum about a 16 pct decrease in the diameter-of-separation. At the same time the slurry throughput would increase about 40 pct.

Direct data available at this time are not consistent enough to permit accurate evaluation. Practically, separation is made finer by increasing pressure, but only big changes in pressure make a noticeable difference. An exception may exist in the treatment of plastic pulps.

Plasticity of Feed

If the pulp solids contain more than 2 or 3 pct undispersed clay or claylike solids, the pulp will usually be significantly plastic. In general, separations on plastic pulps will be from one to many meshes coarser than would be obtained on nonplastic slurries. Increasing pressure is particularly beneficial in treating plastic pulps, because the increasing centrifugal forces thus obtained aid in causing particles to tear their way through the plastic structure of the solids. Therefore the mesh-of-separation may become finer more rapidly than normal with plastic pulps, and at very high pressures, as in the range of 70 psi, the separation may often approach that expected for nonplastic slurries. A plastic pulp may usually be recognized by the paintlike coat of solids covering objects dipped in it.

The separation ranges given in Table IV are valid for the feed condition which comprises dilute suspension of −65 mesh nonplastic solids, fed through the cyclones at a head of 50 ft. The ranges are covered by changing the feed entrance areas and the vortex finder diameters. The separation will deviate from the given ranges with changing conditions as described above.

Sharpness of Classification

Mesh-of-separation does not tell the whole story on classification. The next question is how sharp a classification is produced. How free from fines is the coarse product or underflow?

In considering this question it will be helpful to analyze the mechanism of classification by sedimentation. This is shown most clearly in batch sedimentation. Consider a beaker of slurry, containing solids sufficiently dispersed so that classification and not thickening takes place. Originally, as a result of stirring, all particle sizes are uniformly distributed throughout the slurry. As soon as agitation stops, all particles start settling towards the bottom of the beaker at a rate determined by their size and density, among other factors. Those reaching the bottom settle out. After some given time, the slurry above the deposited solids is poured off or otherwise removed, leaving the deposited solids behind. The deposited solids, together with entrained water, represent the underflow of a classifier. The slurry removed represents the overflow.

There is a certain size of particle which during the sedimentation has just time to settle from the top

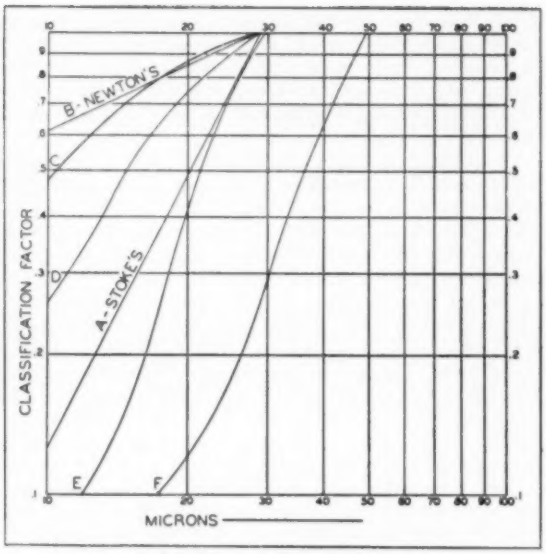

Fig. 2—Classification factor vs micron size.

surface of slurry and deposit on the bottom of the beaker. All particles coarser than this will have already settled out and thus will be totally removed from the overflow fraction. Smaller particles will not reach the bottom or underflow fraction if they start from the top of the pulp, but a fraction of them starting lower in the beaker will reach the bottom in the time allowed. In other words, all particles coarser than the mesh of separation will be collected in the settled fraction and will be absent from the residual overflow suspension. Some fraction of any size range of particles finer than the mesh of separation will also settle into the underflow, the relative quantity depending upon their settling rate.

In addition to particles which travelled into the underflow fraction by settling, there will also be a part of the feed slurry originally occupying the space at the bottom of the beaker which will be invaded later by the settling solids. Some of this is displaced by the settling solids, but some also remains to fill the void spaces between the settled solids. This void filling carries into the underflow an appreciable amount of slimes, or particles so fine as to have insignificant settling rates.

Table IV. Flow and Separation Ranges Expected in Standard Dorr-Clone Units When Fed Dilute, Nonplastic Slurries Comprising —65 Mesh Solids of Various Specific Gravities Suspended in Water, Operating at a Head of 50 Ft, With Various Feed Entrances and Vortex Finders

DorrClone Unit, Diam, In.	Feed Flow, GPM	Separation, 1.35 Sp Gr Solids, Microns	Separation, 2.7 Sp Gr Solids, Microns	Separation, 4.0 Sp Gr Solids, Microns
3	8 to 46	50 to 95	23 to 40	17 to 30
6	43 to 152	62 to 100	28 to 45	21 to 33
12	160 to 550	82 to 128	37 to 58	28 to 44
24	465 to 1460	115 to 150	52 to 67	40 to 50

The underflow from a simple sedimentation classifier, then, comprises three different classes of particles: those coarser than the mesh of separation, or oversize; those having appreciable settling rates but finer than the mesh of separation, or critical sizes; and void filling, which mathematically may be considered to have the same composition as overflow slurry. Although the mechanism has been discussed in terms of simple batch sedimentation, calculations based on its performance have been found valid for continuous sedimentation also.

In practice it has been discovered that the overflow, instead of showing 0 pct plus the mesh of separation, in a screen analysis will show from 1 to 2 pct plus the mesh. Because of shape factors particles are sorted differently by a screen than by a sedimentation process. That is, the screen treats a flaky particle, such as a piece of mica in the extreme case, as much larger than does a sedimentation pool. If the particles were all of exactly the same gravity, and were geometrically similar in shape, then there should be no discrepancy. The shape factors of ordinary milled ores are such as to give about 1½ pct plus the mesh of separation in the overflow. Thus sharpness of classification depends upon the amount of overflow slurry pulled into the underflow as void filling or entrainment, and upon the distribution of critical sizes between the overflow and underflow.

It will be seen that the amount of void filling, and hence the fraction of the feed slimes which appear in the underflow, is determined by the split of water between the two fractions. The less water in the underflow, or to put it another way, the higher the percent solids in the underflow fraction, the more complete the desliming.

In desliming effectiveness, the cyclone can approach an ordinary rake-discharged bowl classifier. It can be substantially more effective than an Hydroseparator. Under comparable conditions, a rake-discharged classifier might give from 22 to 28 pct water in the underflow, a cyclone from 27 to 30 pct water, and a Hydroseparator about 50 pct water. Therefore, a cyclone, if operated with as thick an underflow as feasible, will have in its underflow about the same percent of slimes as a rake-discharged classifier, but only about 60 pct as much as obtained from a Hydroseparator. It can be seen that if desliming is desired, the underflow from a cyclone, or any simple sedimentation classifier, must be run as thick as possible.

In ordinary gravity sedimentation, little control can be exercised over the disposition of the critical sizes. In a cyclone, on the other hand, the split of critical sizes between the underflow and the overflow is strongly affected by operating conditions. A cyclone can be operated to give a separation much sharper than obtained by gravity sedimentation, but it can also be operated to give separations much less sharp.

The following factors have been found to have significant effect upon the distribution of the critical sizes and hence upon the sharpness of classification: percent solids in underflow, size of feed entrance and vortex finder, size of cyclone, and pressure.

The percent solids in the underflow is controlled by the setting of the apex valve, which in turn controls the flow ratio, or gpm underflow divided by gpm feed. A high percentage of solids in the underflow tends to increase the fraction of critical sizes thrown into the overflow. At the same time, as described above, the high percentage of solids gives decreased void filling. Both factors combine to increase sharpness of classification and decrease recovery of feed solids in the underflow. This gives wide variations in the 50 pct separation point but has been found to have relatively little effect on the mesh of separation.

Fig. 2 shows classification factor curves observed for a 3-in. DorrClone unit operated under conditions which were essentially constant except for the underflow percent solids and rate.

The classification factor curves of Fig. 2 are log log plots of equivalent diameter of particles, against the fraction of the particles of that equivalent diameter which settle out during sedimentation as described above. Curve A in Fig. 2 shows what the factor would be if the particles all settled according to Stokes' law. Curve B shows what they would be if the particles all settled according to Newton's law. Note that they are both straight lines on a log log plot. Classification by gravity sedimentation, particularly at fine meshes of separation, will show factors not greatly different from Curve A. Furthermore, the factors will not change significantly with changing underflow rates and percent solids.

Curves C through F show the factors observed for different flow ratios, and hence for different percent solids in the underflow, under conditions shown in Table V.

Note that the curves of Fig. 2 were calculated from runs made with a 3-in. DorrClone unit. In the larger units, the effect is much less pronounced, and the factor curves tend to resemble E of Fig. 2, rather than C, over a much wider range of underflow dilutions. Thus the factor curves, hence also sharpness of classification and recovery of solids in the underflow, are much less influenced by underflow dilution in larger cyclones than in smaller ones.

In all sizes of cyclones, decreasing the size of the feed entrances and vortex finders gives poorer classification and higher recovery of solids to the underflow. Increasing pressure appears to have the same effect.

Table V. Conditions of Tests Showing Effect of Underflow Percent Solids on Classification Factor

Item	All Runs			
DorrClone unit, diam	3 in.			
Feed entrance	0.19 sq in.			
Vortex finder	0.49 in.			
Pressure	37 psi			
Solids in feed slurry	95 pct −325 mesh dolomite			
Solids in feed, pct	43 pct			
Runs	**C**	**D**	**E**	**F**
Solids in overflow, pct	17.7	21.9	26.0	32.4
Solids in underflow, pct	51.8	57.4	62.0	71.1
Underflow gpm divided by feed gpm	.79	.51	.38	.19

The curves of Fig. 2 represent the extreme case. They were calculated from runs on the smallest of the DorrClone units, operating with a small feed entrance and vortex finder, and at a relatively high pressure. The effect will be far less with larger units, and is not particularly noticeable in a 12-in. machine.

Curve F of Fig. 2 is interesting. In all other runs the underflow was discharging in a spray. When there is a spray discharge, the apex orifice is not running full of slurry, but has an air core at the center. As long as a spray discharge is obtained, it is believed that the apex orifice is not overloaded, and no underflow material is backing up in the cyclone to discharge out the overflow. As the apex valve is constricted further and further, a point is reached at which the character of the underflow discharge changes abruptly. The underflow starts flowing in a steady, ropelike stream rather than spraying, and the air core disappears at the apex. This may represent an overloaded condition, wherein underflow is not allowed to escape at the apex as fast as it collects. Curve F shows the results. The mesh of separation becomes coarser. Also a substantial amount of stray or short-circuiting oversize may appear in the overflow.

To obtain a sharp classification, then, the apex valve should be adjusted to give as dense an underflow as possible without rope discharge. This is particularly important when making fine separations, which call for small diameter units, with small fittings and high operating pressures.

Sharp classification is not always desired. Although it has been shown that cyclones are classifiers rather than thickeners, they have been used as substitutes for thickeners where fines can be tolerated in the overflow. In fact, this was one of the original uses for which they were developed by the Dutch State Mines. When substituting for a thickener, it is desirable to collect just as many of the solids as possible in the underflow. Under such conditions the cyclone would be operated to be a poor classifier, throwing critical sizes as much as possible into the underflow rather than into the overflow. Cyclones for this job will normally be as small in diameter as is consistent with operating convenience and will have relatively small ports. They will be operated at high pressures, with as dilute underflows as possible processwise.

Operating Requirements

Interchangeable feed entrance shims and vortex finders give a DorrClone unit considerable flexibility. Without this feature it would not be feasible to adjust cyclone behavior for optimum process performance, or to make the changes in mesh of separation dictated from time to time by process considerations. However, once the unit is set up with the selected fittings, it is essentially a constant flow machine. Means must be provided to maintain the feed flow required by the machine under the chosen conditions. Suitable means have been discussed by Weems.[5] They comprise surge tanks, controlled additions of water where permissible, and recirculation of machine overflow.

Control of the apex valve is often necessary. Small fluctuations in the amount of underflow solids to be discharged will not greatly change the percent solids in the underflow, because in this respect cyclones have a limited self-regulating tendency. To accommodate large changes, if a constant and high percentage of solids in the underflow or a sharp classification is desired, it is necessary to control the apex valve diameter. Where the changes in feed slurry composition are gradual, manual control of the apex valve will often suffice. To handle rapidly changing conditions, an automatic apex control system[6] is available.

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Sampling and Testing of Sinter

by R. L. Stephenson and D. J. Carney

A sampling technique has been developed for procuring a sample of sinter representative of the entire depth of the sintering bed. The sampling method involves the use of an open-bottom metal basket that rides on the grate of the sintering machine and when removed contains a sample of the sintered product. Additional data have been obtained to indicate that the tumbler test is a suitable means of measuring sinter strength.

IN the last few years additional sintering facilities have been installed in both the Pittsburgh and the Chicago district of the United States Steel Co. Since the construction of these sintering plants made possible the use of higher percentages of flue-dust sinter in our blast-furnace burdens, it became important to study means of controlling the quality of sinter to obtain optimum results in the blast furnace.

For controlling an operating process, it is necessary first to establish standards by which the quality of the product can be judged. For sinter, it appeared that an important property was its strength or its resistance to degradation during transportation and charging into the furnace. Consequently work was undertaken to establish a standard for sinter strength that could be used both for controlling sintering-plant operations and for correlating sinter quality with blast-furnace performance.

The first problem in setting up a standard was that of procuring a sample that would be representative of the sinter made under any particular set of conditions at the sintering plant. Since the United States Steel Co. sintering plants discharge the finished sinter either into a large pit or onto a rotary cooler, the sinter becomes inseparably mixed with material sintered 2 hr before or 2 hr afterwards. For this reason the exact identity of the sinter is lost. A sample selected as the cooler is discharged, or as the sinter is removed from the pit, cannot be said to be truly representative of the sinter made at any specific time.

Sampling

The first attempt to procure a sample that would be representative of a specific sinter mix and of specific operating conditions was made by stopping the Dwight Lloyd sintering machine and removing an entire pallet full of sinter. This method, however, proved very difficult to perform and interfered considerably with the operation of the plant. To overcome this difficulty, a sampling method was devised by technologists at South Works enabling them to secure, without interrupting the sintering operation, a sample of about 1 cu ft of sinter, representative of sinter for the full depth of the sintering bed.

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The South Works method involves the use of a steel-frame-work basket. A typical basket is shown in Fig. 1. The basket has been used both with and without crossbars along the bottom. As long as the crossbars are in the same direction as the grate bars on the sintering machine they do not interfere with the sintering process. The basket is set on an empty grate of the Dwight Lloyd sintering machine before it passes under the swinging feed spout, see Fig. 2. When the basket is removed after it has travelled the length of the sintering machine, it contains the sample. Just before the basket is removed, the sinter is scored and chipped to facilitate removal of the sample from the sinter bed. A view of the basket after its removal is shown in Fig. 3. Although the sampling method was originally designed for use on a Dwight Lloyd sintering machine, it can also be used on the Greenawalt type of machine. When used on the Greenawalt-type machine, the basket is placed on the sintering grate before the charging car passes over it, and finally it is removed just before the pan is dumped.

Testing

After a method of obtaining a representative sample of sinter had been developed, the next step was to select a method of measuring its strength. The irregular shape and size of the sinter pieces precluded the use of a simple compression test for determining strength; consequently, the shatter test and tumbler test were investigated. To perform the shatter test, a sample of sinter, approximately 5 lb, is dropped from a hinged-bottom box at a height of 3 ft onto a steel plate. The broken sinter is sieve-analyzed after a specified number of drops. The tumbler test is performed with the use of a standard ASTM coke-tumbling drum. The drum is 3 ft in diam and is equipped with two lifter bars diametrically opposite one another on the inner periphery of the drum. The drum is rotated at a speed of 24 rpm for 200 revolutions, and after tumbling the sample is sieve-analyzed.

To express as single numbers the results of sieve analyses after shattering or tumbling, the method suggested by R. E. Powers[1] was employed. This method involved plotting the size of the sieve openings on a logarithmic scale and the cumulative per cent larger than each sieve on a probability scale as described by J. B. Austin.[2] By interpolating from the plotted data, which in most cases approximated

Fig. 1—Sampling basket, United States Steel Co.

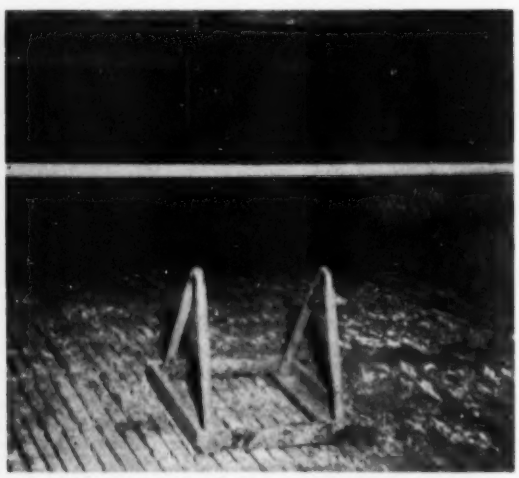

Fig. 2—Sampling basket in place on an empty sintering grate before passing under the swinging feed spout.

Fig. 3—Sampling basket, containing sinter sample, after removal from the sintering bed.

a straight line, the median size could be determined. The median size is taken as the point of intersection of the curve with the 50 pct ordinate.

Results of shatter tests indicated that this test was not severe enough, in that the median size of the sinter after shattering was greatly affected by the starting size of the original sample. On the other hand, the results of tumbler tests revealed that the median size of the sinter after tumbling was almost unaffected by the starting size, and that this test was still selective enough to distinguish clearly between median sizes of sinters of different strength.

Fig. 4 illustrates the effect of starting size on the distribution of particle sizes after shatter testing for 16 drops. From this plot it can be observed that the median size is affected by the size of the pieces selected for performing the test. With a starting size of between ½ and ¾ in., the median size after testing was 0.398 in., whereas with a starting size of between 1 and 2 in., the median size after testing was 0.762 in. Fig. 5 illustrates the effect of starting size on tumbler-test results. In this plot it can be seen that the starting size had little effect on the median size of the sinter after tumbling, for when the starting size was ½ to ¾ in. and when it was 1 to 2 in. the resulting median size was 0.163 in.

Although the shatter test is not severe enough to eliminate wide discrepancies in median size caused by variations in starting size, the tumbler test is severe enough to reduce such discrepancies to a minimum. Furthermore, the tumbler test is not so severe that it destroys the distinction between sinters of different strength. This fact is illustrated in Fig. 6, which shows a plot of tumbler-test results for a sample of weak sinter, medium-strength sinter, and very strong sinter.

To insure getting results that were representative of the full depth of the sintering bed, the entire sample obtained by means of the sampling basket was used as the starting material for the tumbler test. If the sample broke into pieces, all the pieces were used, regardless of their size. Results of tumbling for 10, 20, 50, 100, and 200 revolutions indicated that the median size after tumbling was proportional to the logarithm of the number of tumbler revolutions. This relation is shown in Fig. 7. After 200 revolutions, little effect on the median size of sinter resulted from one or two additional tumbles. Therefore, the 200-revolution test was selected as a standard test because it tends to minimize differences in the severity of handling the sinter between the sintering machine and the testing laboratory. Fig. 8 illustrates typical values resulting from tumbler testing entire samples procured with the sampling basket. A sample of relatively strong sinter had a median size after tumbling of 0.251 in., a sample of relatively weak sinter, 0.184 in., and a sample taken when the sintering was not progressing satisfactorily, only 0.122 in.

The results of this method of testing were easily reproducible in that adjacent samples from the same pallet gave median-size results that agreed with one another within a few thousandths of an inch.

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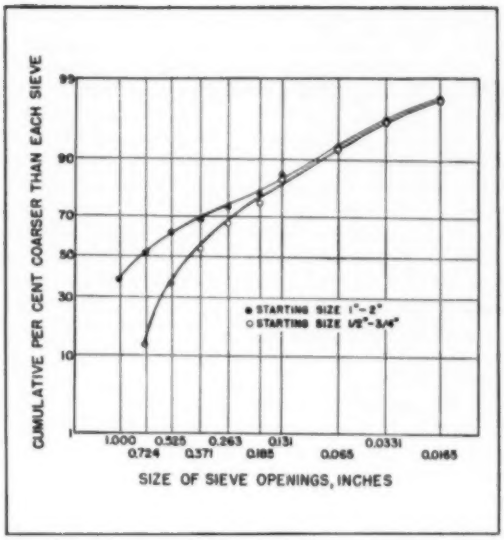


Fig. 4

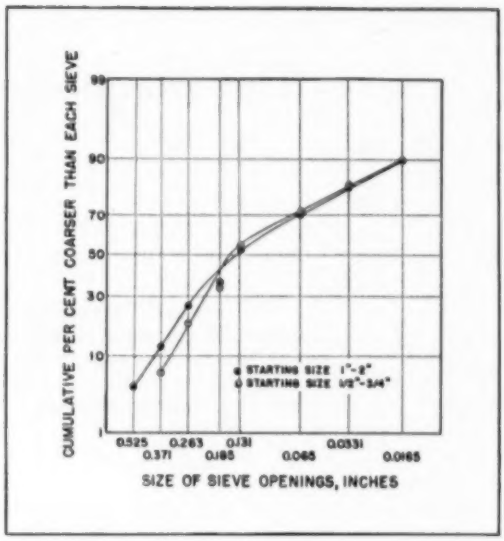


Fig. 5

Fig. 6

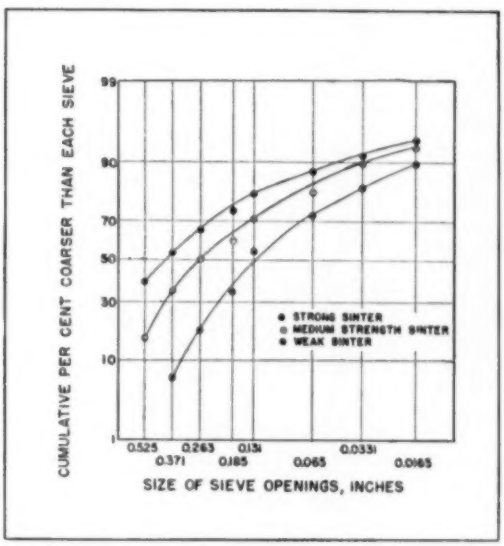


Fig. 7

Fig. 4 indicates the effect of starting size on shatter-test results and Fig. 5 the effect on tumbler test results. Fig. 6 shows the results of sieve analysis of three different samples of sinter after tumbling for 200 revolutions, and Fig. 7 the effect of the number of tumbler revolutions on the median size of sinter samples. Fig. 8 illustrates the results of tumbler tests on samples procured with the sampling basket.

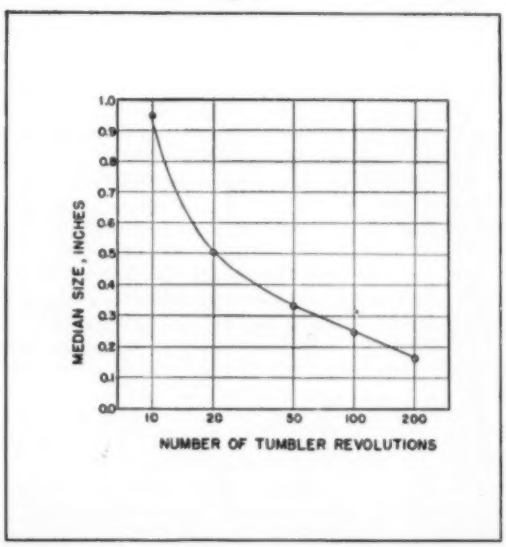
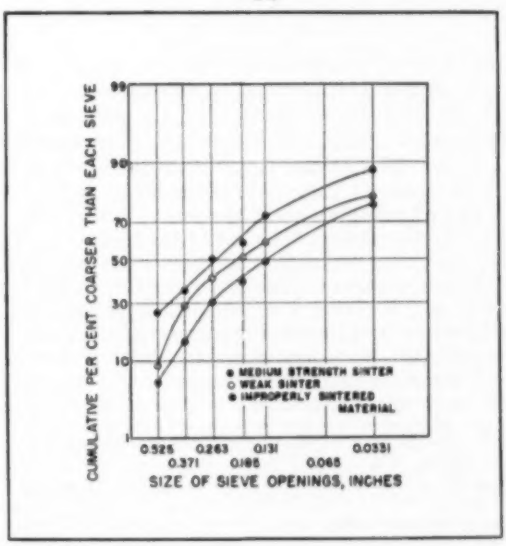


Fig. 8



The Magnetic Reflux Classifier

by Lawrence A. Roe

The magnetic reflux classifier, which utilizes the combined effects of magnetic fields and a hindered settling classifier, is a new tool for determining the quantity and quality of middlings in fine-sized magnetite concentrates. Results are given for processing a typical taconite ore, and a sketch of the apparatus is included.

IN examining magnetite ores and beneficiated products it often becomes necessary to make critical studies of the amount of grinding necessary to produce the desired degree of magnetite liberation. In the past this has been accomplished by laboratory heavy-liquid tests, which provide a method for selectively removing middling particles and free magnetite from various-sized fractions. Examination of the various products under the microscope results in fairly accurate determination of the degree of liberation. The method is quite efficient on sizes coarser than 325 mesh. Thus the heavy-liquid method of middling separation was satisfactory until the advent of present day magnetic taconite studies. When magnetite concentrates ranging from 70 to 100 pct −325 mesh are studied it becomes apparent that older methods of determining liberation size are not satisfactory and that there is need for a new method. For example, some of the low-grade magnetite ores of the Wisconsin and Michigan iron ranges require grinding to 100 pct −325 mesh to produce a magnetic concentrate containing less than 12 pct silica. Examination of concentrates from such ores often reveals that many of the middling particles consist of only very minor proportions of iron mineral.

Thus it becomes important to be able to determine the degree of grinding necessary not only for complete liberation, but also for liberation of only 80, 85, or 90 pct of the total iron mineral content. Actually, complete liberation is never attained, but is often used to designate that degree of liberation necessary for production of high-grade concentrates. A rougher concentrate, produced after elimination of a coarse-sized tailing, can usually be subjected to a second grinding stage and concentrated into a higher grade product than could be produced from the same crude ore with one stage of grinding resulting in the same overall size reduction. This fact adds to the importance of being able to determine partial degrees of liberation on any magnetite ore.

Standard laboratory methods such as heavy-liquid separation, microscopic grain counts, Davis tube magnetic separation, magnetic flocculation, classification, flotation, and others often are not applicable, or are prohibitive because of time requirements when large numbers of fine-sized magnetite samples are investigated. The Davis tube magnetic separator is an efficient tool to use in rejecting the non-magnetic mineral particles from an ore sample. The middlings discarded by the tube separator usually are so low in iron content that they can be considered relatively unimportant in liberation studies. This condition is caused by the extremely high flux density used in the Davis tube. This flux density ranges from four to eight times the flux density produced by most of the powerful commercial machines in use today. Thus the problem resolves itself into a search for a method of selectively removing middlings from Davis tube magnetic concentrates which will be both rapid and efficient.

Those methods showing most promise in the development of a process for isolating middlings from extremely fine-sized magnetic concentrates were flotation and magnetic flocculation. The use of flotation to remove middlings from magnetic concentrates is reported in the literature.[1, 2] The flotation process is effective in removing middlings from a magnetite concentrate, but physical entrapment of fine-sized free magnetite in the silica-bearing froth is an undesirable feature. The flotation method of removing middlings requires time, effort, and precise control of many variables, and does not meet the required degree of middling isolation.

Magnetic Flocculation

Magnetic flocculation has long been resorted to[3-6] in efforts to upgrade magnetite concentrates. One of the new magnetic taconite plants now under construction on the Mesabi Range includes magnetic flocculation in the flowsheet[4] as an accessory process to remove high-silica middlings and free silica which has been mechanically entrapped in magnetite flocs. The use of magnetic flocculation as a laboratory method of making precise separation of middlings was further investigated, since it offered a rapid, simple method of accomplishing the desired result.

Magnetic flocculation involves the subjection of a magnetic concentrate to a strong magnetic field, passing the concentrate in a highly flocculated condition to a hydroseparator or other classifiers of various types, and removing free silica and middlings as overflow products. In an attempt to utilize simple

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magnetic flocculation as a method of removing middlings from a magnetic concentrate, it soon becomes apparent that the method as generally practiced is efficient only in separating those middling particles containing very minor quantities of magnetite. Middling particles containing over 10 to 20 pct by weight of magnetite are strongly attracted to the magnetic flocs and become entrapped in a maze of magnetite particles, all strongly attracted to one another. If the velocity of the upward moving current of water is increased beyond a critical point, the magnetic flocs are torn apart to some extent, and while more middlings are released, a considerable quantity of fine magnetite is released and lost in the overflow product.

The Magnetic-Reflux Classifier

The solution to the problem was found in imposing a controllable magnetic field near the top of a hindered-settling classifier tube. The classifier then became a magnetic-reflux classifier. Fig. 1 is a sketch of one version of the apparatus now in use.

In chemical technology a reflux condenser is defined as "a condenser which continuously returns the condensed vapor to the still." Thus a magnetic-reflux classifier can be defined as a classifier which "continuously recycles or returns magnetically flocculated particles to the upward moving water currents or teeter column."

When a sample of low-grade magnetite concentrate is placed in the classifier tube the middling particles can be removed readily by controlling two variables: intensity of the magnetic field, and velocity of the upward current of water. These variables are the same controlling factors used in most wet magnetic separators, but it must be realized that there is a great difference in the degree of water control in a laboratory hindered-settling classifier tube as compared to water control in any magnetic separator in use today. In some cases it is advantageous to provide additional stirring or dispersing action by use of an AC field which operates intermittently, or a demagnetizing coil near the lower end of the classifier tube.

It has been stated by E. W. Davis of the University of Minnesota Mines Experiment Station that "the tendency for all magnetic particles to remain in the concentrate whether or not they are attached to particles of gangue is characteristic of all direct current magnetic separators when concentrating finely crushed ore. While this tendency can be counteracted to some extent by the use of agitation, with no direct current magnetic machine now in use is it possible to fraction accurately a sample of finely crushed magnetic taconite into products of different iron assay such as can be done by gravity methods using tables, for example."[7] Eketorp also reports on the inefficiency of present DC separation on fine-sized ores.[8]

It is well known that when a classifier is operated to emphasize the differences in specific gravity of the minerals being processed, hindered-settling conditions are preferred. In such cases the classifier not only sizes the materials involved, but concentrates the heavier minerals by a sorting process. This is not true when free-settling conditions are used. Classifier tubes used in laboratory experiments had an ID of 1.80 in. and the constriction plates had holes varying from 0.036 to 0.054 in. in diam. Use of an accurately calibrated manometer on the hydraulic water line assured precise water control to

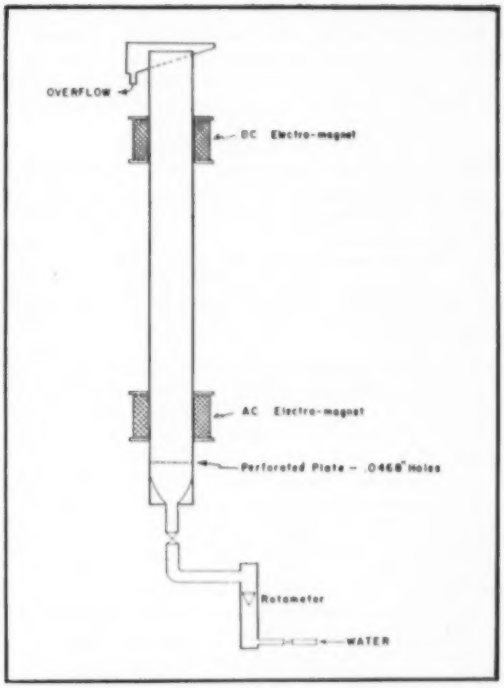

Fig. 1—Sketch of magnetic-reflux classifier for batch tests.

the classifier. High-powered photographers' spotlights were used in studying teeter column conditions in the classifier tube.

The magnetic field imposed on the classifier was provided by a 110-v DC electromagnet. The field of the electromagnet could be varied to produce flux densities from 5 gauss to 300 gauss at the internal surface of the tube wall. This field intensity is very low as compared to the range of 4200 to 14,000 gauss in the Davis tube separator.

Operating Procedures

The use of hindered-settling conditions plus precise magnetic flocculation control results in a laboratory tool capable of separating numerous fractions of magnetite middlings from magnetite concentrates. A typical operational procedure for batch tests with one type of magnetic-reflux classifier used in early tests was:

1—With hydraulic water set at 1700 cu cm per min and field magnet set at high intensity, a 50 to 100-g sample is introduced; all the sample is poured into the tube before the water level reaches the magnetic field level of the tube.

2—Period *a* begins, during which high-silica, low-iron middlings are removed as overflow product.

3—At end of period *a* the magnetic field is adjusted to medium intensity; hydraulic water remains at same flow.

4—Period *b* begins, during which middlings containing nearly equal percentages of silica and magnetite are removed as overflow product.

5—At the end of period *b*, the magnetic field is adjusted to low intensity; hydraulic water remains at same flow.

6—Period *c* begins, during which high-iron, low-silica middlings are removed as overflow product.

7—At end of period *c*, the final magnetite concentrates are withdrawn as classifier underflow.

It is evident that different ores and various degrees of grinding for the same ore may require different magnetic field strengths and different hydraulic water flows, but the proper settings are usually quickly arrived at by preliminary runs.

Test Results

Fig. 2 illustrates the appearance of the flocculated magnetite in the magnetic-reflux classifier tube during each of the three periods described under operating procedures.

The magnetite concentrate being tested was from an ore which had been ground through 100 mesh. Table I gives the chemical analysis of the classifier products.

Table I. Chemical Analysis of Classifier Products —100-Mesh Feed

Product	Wt, Pct	Analysis, Pct		Dist., Pct	
		Fe	SiO_2	Fe	SiO_2
Feed	100.0	54.83	22.86	100.0	100.0
Overflow, period *a*	6.5	20.13	62.00	2.4	19.9
Overflow, period *b*	5.1	29.61	51.38	2.8	13.0
Overflow, period *c*	7.5	43.18	33.04	5.9	12.2
Underflow	80.9	60.29	13.74	88.9	54.9

Table II. Chemical Analysis of Classifier Products, —200-Mesh Feed

Product	Wt, Pct	Analysis, Pct		Dist., Pct	
		Fe	SiO_2	Fe	SiO_2
Feed	100.0	57.98	16.23	100.0	100.0
Overflow, period *a*	5.9	24.86	55.78	2.5	20.3
Overflow, period *b*	6.8	37.24	40.12	4.4	16.8
Overflow, period *c*	4.9	51.02	24.40	4.3	7.4
Underflow	82.4	62.48	10.94	88.8	55.5

From these results it is evident that 88.9 pct of the iron was recovered as concentrate, while 45.1 pct of the silica was rejected in three distinct middling fractions. Photomicrographs of these middling fractions are shown above the photographs of the classifier tube in Fig. 2. The magnetite contents of the three middling fractions were 28, 41, and 60 pct respectively. The degree of flocculation of the respective overflow products gives visual indication of the quantity of magnetite present in each product. Photomicrograph (d) of Fig. 2 shows the highly flocculated classifier underflow product which contained 83 pct by weight of magnetite.

The results listed in Table I show that when a —100 mesh grind was used it was possible to produce a final magnetite concentrate containing 60.29 pct iron and 13.74 pct silica. Present day taconite studies are, in many cases, aimed at producing iron concentrates containing not over 12 pct silica. Another test was made on the ore previously tested, but using a —200 mesh grind. Table II lists these results.

Again, 88.8 pct of the iron was recovered as concentrate and 44.5 pct of the silica was rejected in three distinct middling fractions. However, the concentrate grade was raised to an acceptable figure: 62.48 pct iron and 10.94 pct silica.

For comparison, Table III lists the results of standard Davis tube tests, 1.80 amp, on —100, —200, and —325 mesh ore samples and the results of a Davis tube test at low amperage on —200 mesh ore. The ore was the same material used, after a magnetic concentration step, in the magnetic-reflux classifier tests.

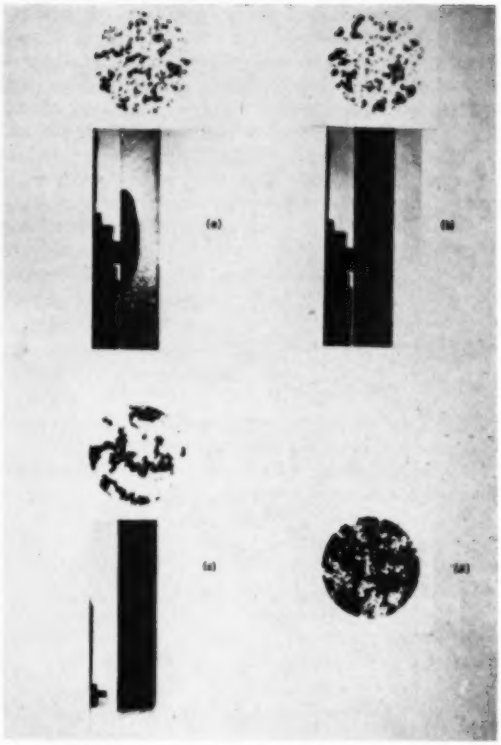

Fig. 2—Appearance of classifier tube during each of the three operating periods *a*, *b*, and *c*. Photomicrographs above the tube show degree of flocculation of the overflow product for each period and the underflow product, *d*. In this particular test the magnetic field was applied to only one side of the tube. Good results were obtained, but capacity was low.

Thus it can be seen that Davis tube tests on extremely fine-grained magnetite ores at either high or low current intensity are not satisfactory for use in liberation studies which have as their objective the rapid determination of an optimum commercial grind. Conventional tube tests show that a —325 mesh grind is necessary to produce a concentrate containing less than 12 pct silica, but no information is obtained regarding the middlings which would be produced in a commercial operation. It was previously shown, by using the magnetic-reflux classifier, that a —200 mesh grind would result in an acceptable concentrate. Also, the three middling fractions in the rougher magnetic concentrate would contain 24.86, 37.24, and 51.02 pct iron respectively.

Table III. Davis Tube Tests

Product	Mesh	Davis Tube Amperage	Analysis, Pct		Dist., Pct	
			Fe	SiO_2	Fe as Magnetite	SiO_2
Feed	—	—	35.82	43.22	95.0	100.0
Magnetic Concentrate	—100	1.80	54.83	22.86	99.0	30.7
Magnetic Concentrate	—200	1.80	57.98	16.23	98.0	22.2
Magnetic Concentrate	—325	1.80	62.89	11.16	98.0	14.4
Magnetic Concentrate	—200	0.20	62.78	11.06	47.9	6.6

The last two fractions, representing 11.7 pct of the weight and 8.7 pct of the iron in the rougher concentrate, would in all probability be reground and recirculated with new feed to the plant. When —200 mesh ore was processed at low amperage in the

Davis tube, an acceptable concentrate resulted, but less than 50 pct of the magnetite reported in the concentrate. When the non-magnetic fraction is reprocessed at successively higher amperages, a series of middling products can be obtained, but the sensitivity of the separation is far too low to enable the segregation of precisely graded middling fractions, and it is difficult to extrapolate how much of the low-silica, high-iron middlings can be added to the 0.20 ampere-concentrate and still not exceed the 12 pct silica figure. At this point it would be well to mention that the Davis tube is a useful tool for studying liberation characteristics of coarse-grained magnetite ores. For the purposes of this paper, coarse-grained ores are those which require grinding not finer than 65 mesh to liberate most of the magnetite. In such ores the extremely fine-sized middling particles are relatively unimportant and thus need not be isolated for study in most cases.

The magnetic-reflux classifier is the subject of a patent application. Interest in the selective separation of middlings from magnetic taconites is running high, and new types of magnetic separators involving principles exemplified by the magnetic reflux classifier may materialize in the future.

Acknowledgment

The original work on the magnetic-reflux classifier was done by the author at the Ore Research Laboratories of Jones & Laughlin Steel Corp., and this work has been continued at the Metallurgical Laboratories of Bjorksten Research Laboratories, Inc., Madison, Wis. Thanks are due both of these corporations for their interest in the problems of magnetic taconite beneficiation.

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Work Indexes Tabulated

by Fred C. Bond

SIX years have passed since the last grindability table was published.[1] In that time the list has been increased with many new tests, and the development of the new Third Theory of Comminution[2] has made possible the reduction of all results to a consistent work index value.

In this tabulation all grindability and crushability test results are given in terms of the work index, with formulas for reversion to the older units if desired. The average values for 57 different classes of materials are listed in Table I. The total number of work index values listed is 1211, and the overall average work index is 14.42.

The complete tabulation, with each test listed alphabetically both under material type and under company name, can be obtained by writing to the Allis-Chalmers Manufacturing Co., Processing Machinery Department, Milwaukee 1, Wisconsin.

The work index *Wi* is the calculated total work input in kilowatt-hours per short ton required to reduce from theoretically infinite particle size to 80 pct passing 100 microns, or to approximately 67 pct passing 200 mesh.

The work inputs required per ton for the same size reduction of different materials are directly proportional to their work indexes, and according to the Third Theory the work input per ton is inversely proportional to the square root of the diameter of the product particles.

Where *W* represents the work input in kilowatt hours per ton, where *F* is the feed size, or diameter in microns of the square hole which 80 pct of the feed passes, *P* the product size, or microns which 80 pct of the product passes, and *Rr* the reduction ratio *F/P*, the work index *Wi* is found from

$$Wi = W\left(\frac{\sqrt{Rr}}{\sqrt{Rr}-1}\right)\sqrt{\frac{P}{100}} \qquad [1]$$

When the work index *Wi* is known, the work input *W* required to reduce from any feed size *F* to any product size *P* is found from

$$W = Wi\left(\frac{\sqrt{Rr}-1}{\sqrt{Rr}}\right)\sqrt{\frac{100}{P}} \qquad [2]$$

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Table I. Average Work Indexes

Material	Number Tested	Average Specific Gravity	Average Work Index
All materials tested	1211		14.42
Andesite	6	2.84	18.25
Barite	7	4.50	4.73
Basalt	3	2.91	17.10
Bauxite	4	2.20	8.78
Cement clinker	14	3.15	13.56
Cement raw material	19	2.67	10.51
Coke	7	1.31	15.18
Copper ore	204	3.02	12.73
Diorite	4	2.82	20.90
Dolomite	5	2.74	11.27
Emery	4	3.46	56.70
Feldspar	8	2.59	10.80
Ferro-chrome	9	6.66	7.64
Ferro-manganese	5	6.32	8.30
Ferro-silicon	13	4.41	10.01
Flint	5	2.65	26.16
Fluorspar	5	3.01	8.91
Gabbro	4	2.83	18.45
Glass	4	2.58	12.31
Gneiss	3	2.71	20.13
Gold ore	197	2.81	14.93
Granite	36	2.66	15.05
Graphite	6	1.75	43.56
Gravel	15	2.66	16.06
Gypsum rock	4	2.69	6.73
Iron ore			
Hematite	56	3.55	12.93
Hematite — specular	3	3.28	13.84
Oolitic	6	3.52	11.33
Magnetite	58	3.88	9.97
Taconite	55	3.54	14.60
Lead ore	8	3.45	11.73
Lead-zinc ore	12	3.54	10.57
Limestone	72	2.65	12.54
Manganese ore	12	3.53	12.20
Magnesite	9	3.06	11.13
Molybdenum ore	6	2.70	12.80
Nickel ore	8	3.28	13.65
Oil shale	9	1.84	15.84
Phosphate rock	17	2.74	9.92
Potash ore	8	2.40	8.05
Pyrite, ore	6	4.06	8.93
Pyrrhotite, ore	3	4.04	9.57
Quartzite	8	2.68	9.58
Quartz	13	2.65	13.57
Rutile ore	4	2.80	12.68
Shale	9	2.63	15.87
Silica sand	5	2.67	14.10
Silicon carbide	3	2.75	25.87
Slag	12	2.83	9.39
Slate	2	2.57	14.30
Sodium silicate	3	2.10	13.50
Spodumene ore	3	2.79	10.37
Syenite	3	2.73	13.13
Tin ore	8	3.95	10.90
Titanium ore	14	4.01	12.33
Trap rock	17	2.87	19.32
Zinc ore	12	3.64	11.56

Eq. 2 is used to find the work input required for any reduction size in dry crushing or wet grinding directly from the listed work index; the work input required for dry grinding is found by multiplying the work for wet grinding by 4/3.

The 80 pct passing size in microns is a convenient term for expressing the fineness of a crushed or ground product, and a convenient base for calculation of the reduction ratio and the work required for reduction.[2] It is readily found by plotting the percent passing size distribution curve.

When the size distribution curves of the feed and product are parallel the reduction ratio remains constant for all particle sizes, and the work index calculated from the 80 pct passing size is equivalent to that calculated on the basis of any other selected percent passing size. Small differences between the slopes of the plotted feed and product lines have only a slight effect on the work index, proportional to the square root of the effect upon the surface areas. However, when the feed has had the fines removed the equivalent feed size is increased and the work index may be considerably in error. In cases where a crusher feed is scalped by passing over a grizzly screen the tonnage and size distribution of the fines removed are rarely known, and it is preferable to consider the feed to the grizzly as being equivalent to the feed to the crusher and to calculate the work index on this basis.

The listed work index values calculated from laboratory grindability tests apply directly to a wet grinding overflow type of rod mill 6 ft in diam in open circuit, and to a wet grinding overflow type of ball mill 7½ ft in diam in closed circuit with a rake classifier at 250 pct circulating load, and with 80 pct or more of the feed passing 4 mesh. Correction factors should be applied for oversize feed and other operating conditions which change the grinding efficiency. When work index values at several different product sizes are available the value nearest the actual product size should be used. The work index represents the total work input to the reduction machine necessary in installations of average efficiency.

Where C represents the impact crushing strength[1] in foot-pounds per inch of thickness, and S is the specific gravity, the work index is

$$Wi = 2.59\,C/S \qquad [3]$$

and the impact crushing strength is found from the listed work index by

$$C = Wi\,S/2.59 \qquad [4]$$

Where Grp represents the rod mill grindability[1] in net grams produced per revolution of the test mill which pass a sieve opening of P_1 microns, the work index is found from

$$Wi = \left(\frac{16.6}{Grp^{0.625}}\right)\sqrt{\frac{P_1}{100}} \qquad [5]$$

and the rod mill grindability is found from the listed work index by

$$Grp = \left[\left(\frac{16.6}{Wi}\right)\sqrt{\frac{P_1}{100}}\right]^{1.60} \qquad [6]$$

Where Gbp represents the ball mill grindability at P_1 microns the work index is found from

$$Wi = \left(\frac{16}{Gbp^{0.820}}\right)\sqrt{\frac{P_1}{100}} \qquad [7]$$

and the ball mill grindability is found from the listed work index by

$$Gbp = \left[\left(\frac{16}{Wi}\right)\sqrt{\frac{P_1}{100}}\right]^{1.22} \qquad [8]$$

The work index can be calculated by Eq. 1 from commercial crushing or grinding data or from pilot mill tests, and compared with the laboratory work index to obtain the relative mechanical efficiency.

The capacity of any size-reduction machine over its normal range of feed and product sizes is proportional to $\sqrt{Rr \times P}/(\sqrt{Rr} - 1)$. In cases where the capacity is found to vary more than this amount some condition causing inefficient operation should be suspected. These may include packing in a crusher, oversize feed or improper ball and rod sizes in a tumbling mill, and coating in dry grinding.

References

[1] Fred C. Bond: Standard Grindability Tests Tabulated. *Trans.* AIME (1949) **183**, p. 313.

[2] Fred C. Bond: The Third Theory of Comminution. *Trans.* AIME (May 1952) **193**, p. 484.

[3] Arthur F. Taggart: New Units of Crusher Capacity and Crusher Efficiency. *Trans.* AIME (1943) **153**, p. 319.

Geophysical Case History, Fredericktown Lead District, Missouri

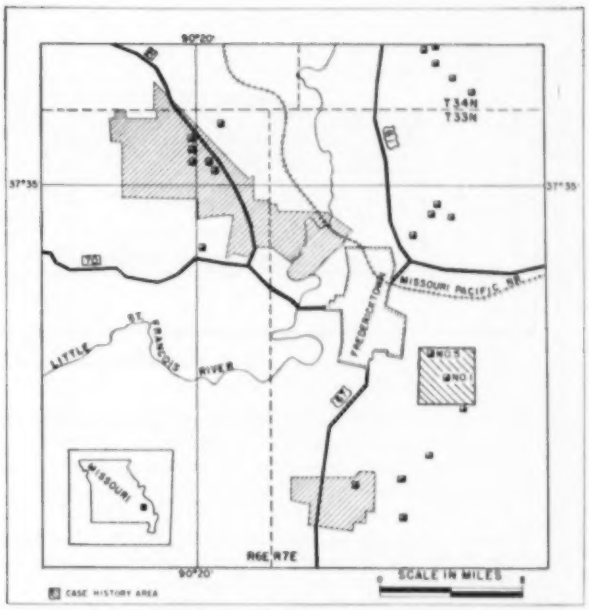

Fig. 1—Index map showing location of case history area, Fredericktown lead district, Missouri.

by Harold Powers, LeRoy Scharon, and Carl Tolman

THIS paper presents geophysical and subsurface data observed in the vicinity of Shafts No. 1 and and 5 of the National Lead Co. lead mines at Fredericktown, Madison County, Missouri, see Fig. 1. The area selected is an excellent example to illustrate the close correlation between geophysical data and geological facts as revealed by drilling and mining. Such close correlations make geophysical applications appropriate to the development of known orebodies as well as the exploration for new orebodies.

In the past, magnetic and shallow electrical resistivity observations were made over limited areas in the Fredericktown lead district by the Missouri Geological Survey, with favorable results. The first extensive magnetic and electrical resistivity surveys were started by the U. S. Bureau of Mines in May 1945 in two areas located northwest and southwest of Fredericktown, as shown in Fig. 1. The results of these surveys were reported in April 1948 at Denver, Colo. Late in 1945 spontaneous-polarization surveys were made over the property of the St. Louis Smelting and Refining Co. at Fredericktown by Sherwin F. Kelly Geophysical Services, Inc. Aeromagnetic surveys were completed in 1948 over the Fredericktown quadrangle by the U. S. Geological Survey.

Since 1945 geophysical surveys have been an important part of exploration programs in the Fredericktown district. Such methods, magnetic and electrical resistivity, are now serving as guides in the further development of known orebodies.

On an irregular pre-Cambrian igneous erosion surface Lamotte sandstone of upper Cambrian age was deposited, partially filling the topographic basins, see Fig. 2. Above and conformable with the Lamotte sandstone lies the Bonneterre formation, a dolomite with locally interbedded thin dolomitic limestones and shales at its base. This formation also only partially filled the topographic basins. The Bonneterre was overlain by the Davis, Derby, Doe Run, and Potosi formations, which are principally dolomite.

Uplift, tilt, and slight warping has allowed erosion and weathering to remove most of the sediments overlying the Bonneterre, and in many places, not far from the mine, the Lamotte sandstone is exposed. This erosion has also exposed areas of pre-Cambrian igneous rock.

The Lamotte sandstone, having been deposited on an irregular pre-Cambrian surface, is extremely variable in thickness and ranges from zero where it laps the pre-Cambrian rocks to 250 ft thick in the deeper basins. In the vicinity of the mines the Bonneterre dolomite ranges from zero where it wedges out against the pre-Cambrian igneous rocks to 400 ft thick.

The ore deposits occur as a bedded disseminated type[1] and are chiefly composed of galena and pyrite with minor amounts of chalcopyrite and siegenite. Heavy pyrite and marcasite mineralization may be found throughout the Bonneterre formation. However, there is little or no apparent association of these iron sulphide bodies lying above the ore hori-

H. A. POWERS, Member AIME, is Mining Geologist, National Lead Co., Fredericktown, Mo. H. L. SCHARON and C. TOLMAN, Members AIME, are Associate Professor of Geophysics and Professor of Geology, respectively, at Washington University, St. Louis, Mo.

Discussion on this paper, TP 3507L, may be sent (2 copies) to AIME before May 31, 1953. Manuscript, Feb. 20, 1952. New York Meeting, February 1952.

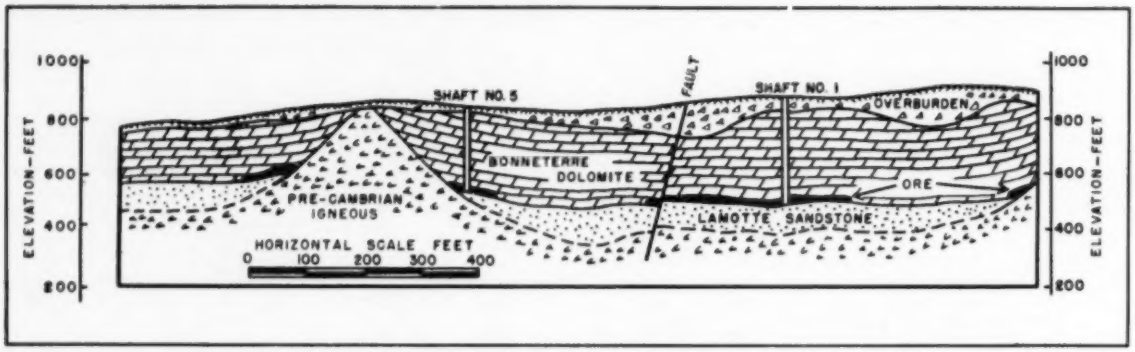

Fig. 2—Generalized geologic section, Fredericktown lead district, Missouri.

zon and the ore deposits. Economic mineralization is characteristically confined to the lower 20 to 25 ft of the Bonneterre dolomite. The average depth from surface to this favorable zone is 300 to 400 ft.

For prospecting and development purposes, the orebodies can be divided into two types. One type characteristically occurs near the Lamotte sandstone pinch-out against the pre-Cambrian igneous. The other type is in areas of unusual Lamotte sandstone highs and lows. The erratic contour of the Lamotte sandstone may have been caused by sand being deposited on an irregular pre-Cambrian surface, by the formation of sandbars at the time of deposition, or by tectonic disturbances.

The host rock near the Lamotte sand pinch-out line has been favorably prepared by a series of fractures including joints and minor faults which parallel the Lamotte pinch-out line and often coincide with it. The host rock in areas of Lamotte sandstone highs and lows is structurally prepared in much the same manner. Within the area at least one fault is believed to displace the pre-Cambrian igneous rock.

There are two known localized sites for ore deposition within the immediate mine area, one being the lower 20 to 25 ft of the Bonneterre dolomite near the Lamotte sandstone pinch-out line and the other being the lower 20 to 25 ft of the Bonneterre dolomite in areas of Lamotte sandstone highs and lows. Both localized areas appear in some measure to be associated with the pre-Cambrian topography.

An area of approximately 370 acres in the vicinity of Shafts 1 and 5 was chosen to ascertain the practicability of using geophysical methods as aids in the development of ore reserves. Magnetic, spontaneous polarization and resistivity surveys were conducted as described in detail below.

Although spontaneous polarization measurements did not prove practical in assisting in the location of new orebodies or extending known ones in the Fredericktown district, magnetic and resistivity surveys have led to a more intelligent development of ore reserves by actual drilling. Magnetic surveys revealed the presence of buried igneous knobs and ridges or the subsurface extensions of those already known. After the location of such igneous masses had been established, electrical resistivity measurements helped in determining the depth to the igneous surface, the depths to the contacts of the various sedimentary horizons involved, and the location of zones of fracture.

Magnetic Surveys: Measurements of vertical magnetic intensity in the area were done on a 100-ft

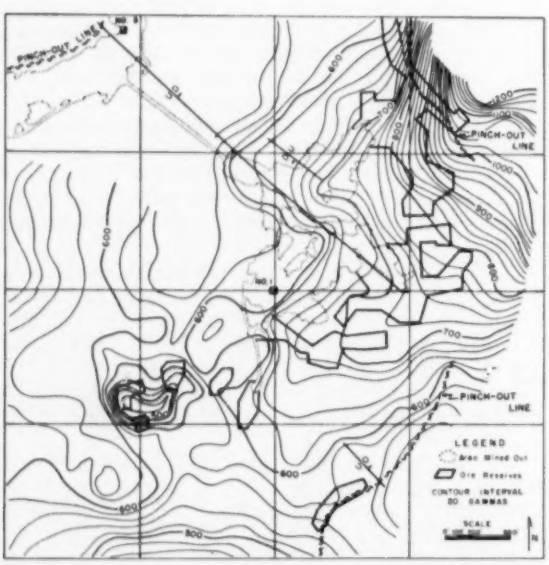

Fig. 3—Vertical intensity magnetic map showing mined-out areas and ore areas.

Fig. 4—Self-potential map showing locations of electrical resistivity depth stations.

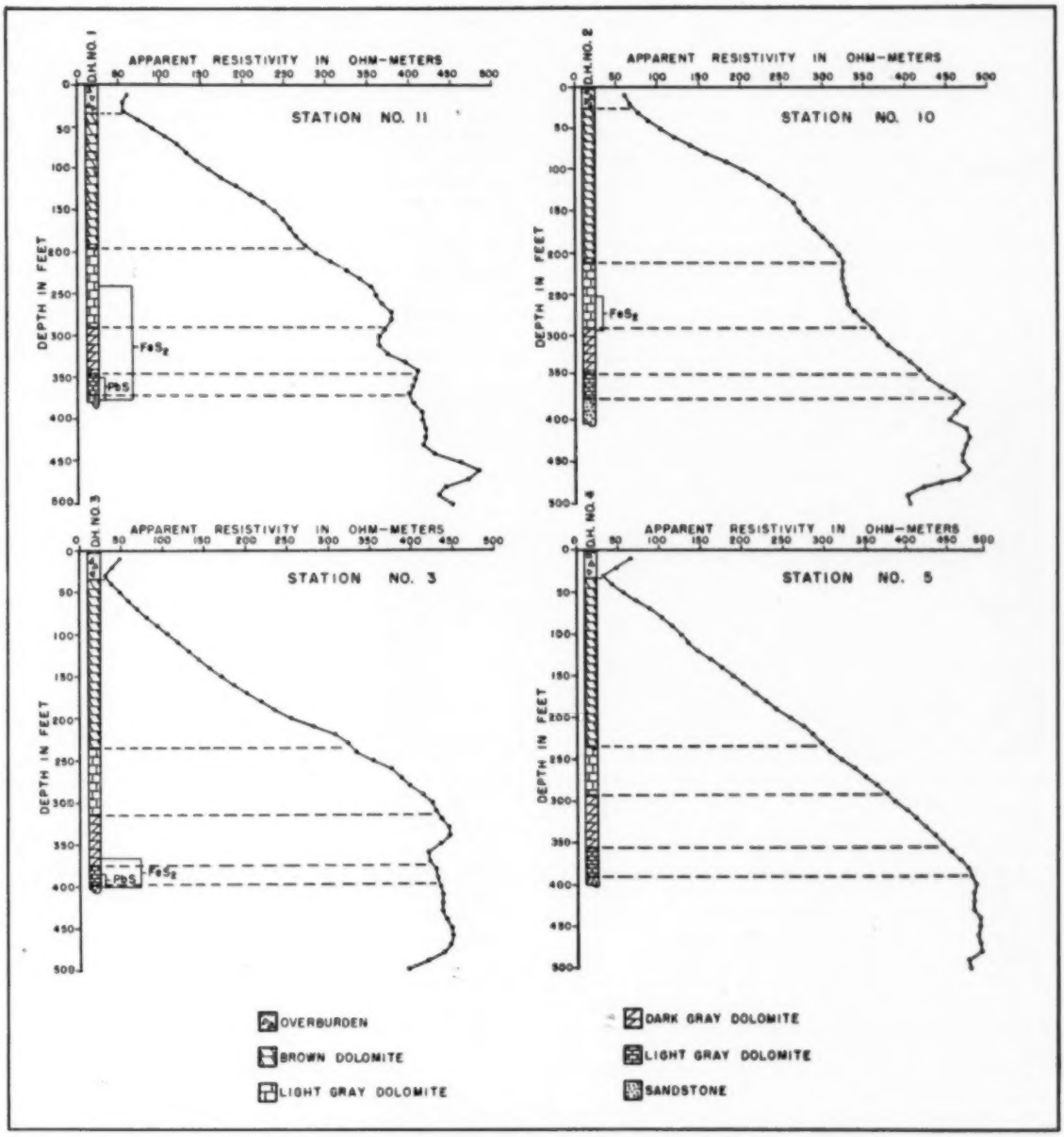

Fig. 5—Apparent resistivity profiles correlated with drill log data.

grid using an Askania-type vertical intensity magnetometer with a scale constant of 20 gammas. From these observed data an isogam map, Fig. 3, having a contour interval of 20 gammas has been prepared and superimposed on a structural map showing mined ore, proved ore, and the pinch-out line. Measurements could not be made in the vicinity of Shaft No. 5 because of surface interference.

Considerable variation in magnitude and trend of the magnetic lines appears to be correlated with known geological facts. For example, a magnetic maximum of 1200 gammas is associated with the buried igneous mass northeast of Shaft No. 1, decreasing in a southwest direction toward the shaft. This suggests that the surface of the igneous mass dips southwest in the form of a ridge and conforms with a sand high in the same area. Then too, the isogam lines appear to be associated with the major fault located north of Shaft No. 1. This association is best observed by following the 600-gamma contour line. It trends southwest to the fault, which it parallels southeast for 900 ft and then southwest past Shaft No. 1. Ore was of good grade in the vicinity of the fault. Furthermore, the magnetics disclosed a likelihood of structural features in the igneous basement southwest of Shaft No. 1 centering around the magnetic low of 480 gammas. In this area it is observed that the 600-gamma contour lines block this low on three sides; the existence of faulting similar to that mapped in the mined-out area to the northeast is therefore assumed. On the basis of this interpretation and the fact that ore was mined in relation to the major fault it was recommended that drilling be concentrated around the flanks of

this magnetic low. Such drilling has revealed three areas of ore grade. A sand low which also indicates faulting exists in approximately the same area. Finally, the magnetics reveal a high in the basement rock in the extreme southwest corner of the area. Subsequent drilling revealed a sand high, but only low grade mineralization. To summarize, it can be said that in detail magnetics are an indirect guide for delineating areas in which to concentrate drilling.

Spontaneous Polarization Surveys: Late in 1945 Kelly[2] made a spontaneous polarization survey which included this area. This method depends upon measuring at the ground surface weak electrical currents generated by sulphide deposits. Over such deposits potentials of a few tens to several hundred millivolts may be observed. In this area readings were made along east-west parallel traverses spaced at 200-ft intervals. Readings were usually taken at 50-ft intervals along these lines, although in some places this interval was reduced to 25 ft. From these observed data a self-potential map with a contour interval of 25 mv was prepared by Kelly and is reproduced in Fig. 4.

A striking anomaly of 100 mv is observed near the No. 1 Shaft and in the general vicinity of the No. 1 orebody. However, it is believed that this anomaly is due to surface conditions. There was no marked indication of other areas over which drilling could be recommended. It may be observed that any anomaly measured by this method is produced by the electro-chemical activity of pyrite located in the oxidizing zone and not by the sulphides in the ore horizon, which lies below the water table.

Electrical Resistivity Surveys: At the time of the electrical surveys the orebodies now established south of Shaft No. 1 were not known. The southern pinch-out line had been fully explored by drilling which revealed the orebody on the southern end. In the area between the southern pinch-out line and the mined-out area south of Shaft No. 1 electrical resistivity measurements were made to guide drilling rather than to drill on a grid system.

The electrical resistivity method, using the Gish-Rooney type of equipment[3] and the Wenner[4] electrode configuration, was employed. Three traverses were established, see Fig. 4, with 32 resistivity station centers at 100-ft intervals. Each station was observed to a depth of 500 ft, a measuring interval, a, of 10 ft being used.

The electrical data was plotted as a function of the measuring interval. Two sites were recommended as most likely to bear conditions suitable for mineralization. However, other drilling was also carried on to test the interpretations fully.

Four apparent resistivity profiles representative of the types measured, with resultant drilling data, are shown correlated in Fig. 5. Drilling was recommended and carried out at these locations and the data compared with the observed resistivities. These calibrations, which are significant, were used to interpret the entire electrical survey. Very definite correlations are shown at the overburden-dolomite and dolomite-sandstone contacts. Intermediate zones of the Bonneterre are not readily correlated. Only two holes, 1 and 3, logged good ore grade mineralization. These ore zones are reflected by the decrease of resistivity, Stations 11 and 3, just above the sandstone in the lower light gray member of the Bonneterre. At Station 10 a decrease in resistivity occurs at a depth of 210 ft. This decrease is too shallow to represent galena mineralization but probably reflects the heavy iron sulphides and fractured ground encountered in drilling. A similar decrease was measured at Station 11 at a depth of 290 ft, again corresponding to heavy iron sulphide mineralization and fractured rock. Where the dolomite is fairly uniform, Station 5, the apparent resistivity profile shows a continuous rate of increase until it reaches the sandstone contact. Drilling at Station 5 and in areas of similar resistivity curves revealed neither iron nor lead sulphide mineralization. Hence the electrical resistivity results do indicate areas which are structurally favorable for mineralization and can be used to guide further drilling. But the method does not directly reveal galena mineralization.

This electrical resistivity work not only established the limits of the orebody south of Shaft No. 1 but also resulted in a considerable saving of drilling costs by eliminating the need for pattern drilling.

Conclusions

The use of magnetic and electrical resistivity prospecting in the Fredericktown lead district has long been established as part of the program for extending and exploring new ore zones. When taken in close coordination with geologic data available, general geologic conditions can be recognized and used intelligently to develop orebodies further. Considerable savings in costs are effected through drilling guidance and reduction of the drilling footage normally required. It is understood that neither magnetic nor electrical resistivity methods give a direct indication of ore, but when properly used, in the light of good geologic mapping, they do indicate conditions favorable for mineralization.

References

[1] Jack A. James: Geologic Relationships of the Ore Deposits in the Fredericktown Area, Missouri. Missouri Geological Survey, *RI* No. 8, 1949.

[2] Sherwin F. Kelly: Report to St. Louis Smelting and Refining Company, 1945.

[3] O. H. Gish: Improved Equipment for Measuring Earth-Current Potentials and Earth-Resistivity. *Bull.*, National Research Council, Vol. 2, Part 2 (November 1926) p. 86.

[4] Frank Wenner: A Method of Measuring Earth Resistivity. *Bull.*, National Bureau Standards, Vol. 12, No. 3, 1916, pp. 469-482.

Correction

In the October 1952 issue: TP 3385F. Application of Coarse Coal Magnetite Separators in an Existing Circuit. By V. D. Hanson, W. K. Heinlein, and J. M. Vonfeld. P. 985, col. 2, par. 4, should read: Magnetite consumption for 1951 at Champion was 0.8 lb per ton of feed, or, since the combined recovery is 53 pct, the consumption per clean coal ton was 1.5 lb.

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aime NEWS

AIME Again Sponsors Student Prize Contest

The AIME will again sponsor a Student Prize Paper competition during the current academic year.

The competition is open only to Student Associates of the Institute. Two divisions are set for the contest, graduate and undergraduate.

The AIME will cooperate with Sections and Student Chapters wishing to hold preliminary local competitions to determine representatives in the national contest. In the national contest, three cash prizes of $100 each will be awarded to the three top papers, while two $100 awards will be given in the graduate division. Students in Local Sections not holding competitions are eligible to enter the national competition directly. However, papers must be submitted to local Section Secretaries for screening.

All papers submitted must represent as completely as possible original work of the author in interpreting experimental results and in writing the paper. Illustrations must be suitable for reproduction.

A special board of judges will choose winners of the national competition. The chairman will present awards at the Annual Meeting.

Announcement was made at the 1953 Annual Meeting of the winners of the National Student Prize Paper Contest. Each author receives $100.

Graduate Division: Bruce W. Gilbert, *Shore Scleroscope Hardness Tests Made on Mohs' Scale Minerals from Talc to Quartz, Inclusive*, entered by Chicago Section. William Fred Hofmeister, *Stress Corrosion Cracking of Plain Carbon and Low-Alloy Steels in Sulphuric Acid*, also entered by the Chicago Section.

Undergraduate Division: Benjamin C. Allen, *A Study of Carbide Solution in Austenite of a 6-4-4-2 High-Speed Steel*, entered by the Philadelphia Section. Marvin A. Kunde, *The Agglomeration of Taconite Concentrate*, entered by the Colorado Section. David F. Drinkhouse, *An Abandoned Iron Deposit, Durham, Pa.*, entered by Lehigh Valley Section.

Don Fernando C. Fuchs, 50 year member, addressed some 30 persons who attended a recent meeting of the Lima, Peru Section. On the extreme left is James S. Young, 57 year member. Following Mr. Young are Don Ernesto A. Baertl, Vice Chairman; Mr. Fuchs; C. W. Westphal, Chairman; and Walter E. King, Secretary-Treasurer.

AIME Members May Express Div. Choice

Every member of AIME is entitled to indicate *one* division as his primary interest and to belong to one or more of the 10 professional divisions serving the membership. By notifying AIME headquarters which division or divisions you wish to be affiliated with you are automatically placed on the mailing list of these divisions and thus will receive special division communications, notices, and news letters, if any. Such affiliation is now shown in the geographical section of the Directory. More than one third of the membership have made no selections. The 10 professional divisions of AIME are as follows: Mining, Geology and Geophysics Div., Coal Div., Industrial Minerals Div., Minerals Beneficiation Div., Extractive Metallurgy Div. (nonferrous), Iron and Steel Div. (production), Institute of Metals Div. (ferrous and nonferrous physical metallurgy), Petroleum Div., Mineral Industry Education Div., and the Mineral Economics Div.

Write to the Secretary, AIME, 29 West 39th St., New York 18, N. Y., advising of the divisions with which you wish to be affiliated indicating *one* division as your primary interest.

Income Shows Surplus

AIME income for 1952 totaled $593,381, compared with $522,076 in 1951, and expenses amounted to $575,315, compared with $514,067 in the previous year. The surplus was therefore $18,065. Of this amount, $10,000 will be returned to the Special Fund for Publications, which was almost wiped out during the years of deficits, and the rest credited to surplus account. All of the expense for the Directory, which was not issued in 1951, was charged in the 1952 accounts though not actually mailed until 1953. This totaled $20,542, including the entire labor expense of the Flexoprint installation which permitted the Directory itself to be printed for about half of the cost heretofore incurred. It is planned to publish the Directory annually hereafter, beginning about June 1954; a supplement will be issued in June 1953, with the names of the committee personnel and Local Section, Division, and Branch officers for the current year.

On a Branch accounting basis, the Mining Branch had a surplus of $27,075; the Petroleum Branch a surplus of $6850, and the Metals Branch a deficit of $15,860, largely because of the relatively large volume of physical metallurgy papers.

Cabinet Selections Reflect Well on AIME

Selection by President Eisenhower of two notable members of the AIME for cabinet posts, as Secretary of the Treasury and Secretary of Defense, is not only a great compliment to George M. Humphrey and Charles E. Wilson, but to the AIME. Mr. Humphrey joined the Institute back in 1922, shortly after he had joined the staff of the M. A. Hanna Co. as an attorney. Those who attended the 75th Anniversary Celebration of the AIME at the Waldorf-Astoria in March 1947 will recall that he was awarded the Charles F. Rand Medal for mining administration at the Annual Banquet. (See Personals, P. 224, for thumbnail sketch of Charles E. Wilson, Secretary of Defense.)

The AIME has been fortunate in having several of its members serve in cabinet posts during the last generation. Herbert Hoover, of course, who served as Secretary of Commerce from 1921 to 1928, and as President of the United States for the next four years, was our most conspicuous representative. Among his cabinet appointments was Ogden L. Mills as Secretary of the Treasury, who, like Mr. Humphrey, was an AIME member. In 1933, President Roosevelt named another AIME member as Secretary of War—George H. Dern, who had been Governor of Utah. And in 1940 Charles Edison was named Secretary of the Navy.

AIME members are not unknown in the halls of Congress and in important diplomatic posts as well.

All have rendered conspicuous service to their country, and should be an inspiration to other members of the profession to aim for the highest rewards of citizenship.—E.H.R.

Student Conclave Held in New York

More than 175 students flocked to the student conclave held at Columbia University in New York City under the sponsorship of the New York Section, February 6 and 7.

The first day's activities began with a luncheon at Columbia's John Jay Hall. Plato Malozemoff, Chairman of the New York Section welcomed the students, while Andrew Fletcher, AIME President, addressed the gathering. Mr. Fletcher commented on the meaning of the Institute to the Student Associate, technical training, and other topics.

The remainder of the afternoon was taken up with a *Careers Conference*. Discussing the future with the students were: Sam Rosch, Anaconda Wire & Cable Co., on *Physical Metallurgy;* Louis Raymond, Ford, Bacon & Davis, *Mining;* James Kemp, Anaconda Copper Mining Co., *Extractive Metallurgy;* Thomas Gillingham, U. S. Atomic Energy Commission, *Geology;* and Alvin W. Knoerr, Engineering and Mining Journal, *Opportunities off the Beaten Track.*

At dinner, held at the Brass Rail, two motion pictures, *Pigs and Progress*, and *Yellow Magic*, were shown.

Possibly the most outstanding feature of the two-day conclave took place February 7, when students split into groups for various field trips. Visits were: Underground trip No. 1, New Jersey Zinc Co.; Fairless Steel Plant; underground trip No. 2, Warren Pipe & Foundry; underground trip No. 3, Alan Wood Steel Co.; and a trip to a copper refinery.

E. Hayes Addresses Columbia Subsection

Earl T. Hayes, chief of the physical metallurgy branch, U. S. Bureau of Mines, Region II, Albany, Ore., addressed the Columbia Subsection on *Zirconium Metallurgy Today* at the group's first meeting of the new year.

Speaking at the Spokane Hotel, Spokane, Wash., he told the group that the Bureau of Mines plant at Albany, Ore. is the only place where zirconium is being extracted in the nation, at a rate of more than 100,000 lb yearly.

He described the process which involves exclusion of air in the manufacturing and melting of zirconium. Basically it involves reduction of zirconium chloride with molten magnesium, under an atmosphere of helium, Mr. Hayes said.

Mr. Hayes also stated that about 20,000 to 25,000 tons of zircon sands are being used annually by U. S. industry for ceramics, refractories, and chemicals.

Around The Sections

• R. G. Tyler, professor at the College of Engineering, University of Washington, addressed the **North Pacific Section** on *Observations on Engineering in Israel* at a recent meeting of the group. A. H. Mellish and G. E. Sigler, Chairman and Secretary-Treasurer of the Section respectively, are new officers.

• A summary of AIME affairs was presented to the **Ohio Valley Section** recently by Past-President M. L. Haider. Meeting at the Beechwold Restaurant in Columbus, the 70 members attending the meeting also heard Mr. Haider discuss supply and demand in the oil industry. At another meeting the group heard Iver Igelsrud, Battelle Memorial Institute on *Mineral Resources of the Ocean.* At its most recent meeting, the **Ohio Valley Section** heard G. C. Gambs, Pittsburgh Consolidated Coal Co. speak on *The Production of Synthetic and Liquid Fuels from Coal.* Chairman T. W. Reed presided at all sessions.

• *Taconite Processing* was the subject of E. W. Davis, head of the Mines Experimental Station, University of Minnesota, at the February meeting of the **Chicago Section.** The gathering was held at the Chicago Bar Association with 164 persons attending. Shadburn Marshall, research associate, U. S. Steel Co., is scheduled to address the group on *Manganese Recovery From Slags* at the March meeting.

Establish Southern California Subsection

Basil Kantzer, Chairman of the Southern California Section, requested the Board's permission for establishment of the Southern Sierra Subsection of the Southern California Section, which was granted. The San Francisco Section had indicated its willingness to release all of Mono and Inyo Counties and that part of Kern County east of the Sierras north of Mojave, and part of San Bernardino County north and northeasterly of Highway 466.

AIME Contributes To ECPD Engineering Study

AIME is contributing $400 to Engineers Council for Professional Development for a special study of the adequacy and standards of engineering education. Other constituent sessions of ECPD are also contributing their share, and the Engineering Foundation is supplying $3000.

New Journal Procedure

A new procedure has been developed at Institute headquarters relating to the despatch of journals to new members. The Admissions Committee meets once a month, normally about the second Wednesday, and reviews all applications that have been received since its last meeting. It recommends for election most of those who have applied. A week later they are elected by the Board. Within a week or ten days thereafter the issue of the journal for the month in which the applicant was elected is mailed to him. Thus an applicant may receive a copy of the current journal of his choice anywhere from about 3 to 7 weeks after mailing his application from any point in the United States.

Only three issues of a journal are sent unless the applicant accepts election by paying his dues.

Texas Shows Growing Interest in Institute

As an indication of the rapidly growing interest in the AIME among oil men, and of the activity of membership committees in Texas, the Southwest Texas and the West Central Texas Local Sections monopolized the awards in the three membership contests of 1952. The President's Banner, for display at Local Section meetings, was won by the Southwest Texas Section for sending in the most new applications from April 1 to December 31 in proportion to its membership at the earlier date. Its membership was 260 and it turned in 87 applications, or 33.5 pct. The same Section won the gavel for sending in the most applications regardless of membership. Runners-up in the Banner contest were: Permian Basin, 25.0 pct; North Texas, 23.2 pct; Mid-Continent, 20.2 pct; and Lehigh Valley, 18.2 pct. Runners-up in the gavel contest were Mid-Continent, 79 applications; Chicago 72; Permian Basin, 71; and New York, 65.

The Petroleum Branch contest, which was for the largest number of applications sent in during the last four months of the year, based on membership, was won by the West Central Texas Section, with a percentage of 28.9 based on a membership of 45. The prize was a projector.

Speaker Named For Northwest Conference

George Vincent, public relations officer for the Aluminum Co. of Canada, Ltd., will address the metals luncheon, Saturday, May 16 at the second Pacific Northwest Metals & Minerals Conference, May 14 to 16. The Kitimat-Kemano project now under development in British Columbia will be the subject of his address. One of the largest Canadian undertakings ever to be financed by free enterprise, the Alcan project includes a major hydroelectric development, an aluminum smelter, and construction of a new community having a projected ultimate population of 50,000.

All events during the Conference will take place at the Benjamin Franklin Hotel, Seattle.

Dues Reminder

The January, February, and March issues have been sent to members whose dues are unpaid, but that they will not receive further issues of the journal after March until their dues have been paid up to date.

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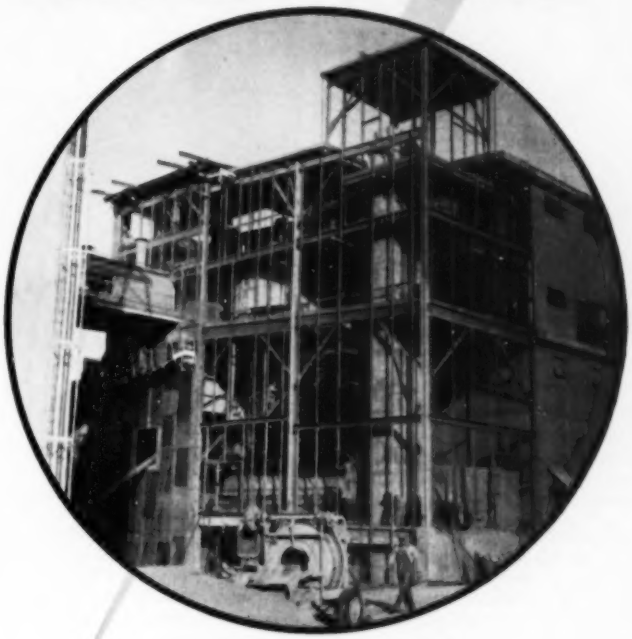
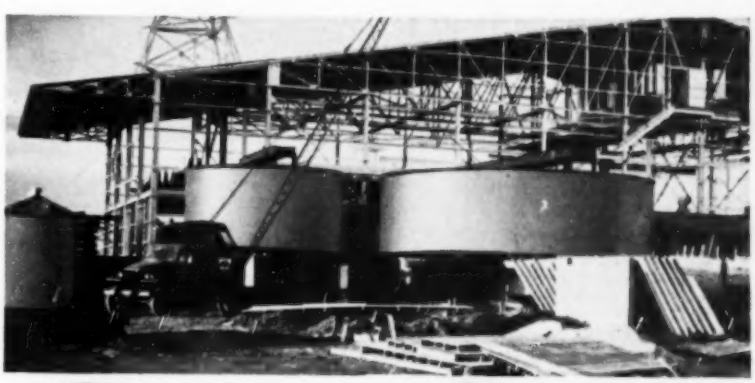


● No matter what size of plant you require... from a multi-million dollar new plant down to alterations totaling but a few thousand, we suggest an early conference.

Stearns-Roger offers the "plus value" of wide experience, over more than two-thirds of a century and in many projects in the fields of metallic and non-metallic processing.



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Personals

LARRY NAHAI

Larry Nahai, chief, Near East Branch, Region IX, U. S. Bureau of Mines, left Washington recently for a four-month observation tour of important mining areas in South Asia and the Near East.

H. C. Allington has been elected vice-president in charge of sales of the eastern div. of the Colorado Fuel & Iron Corp. Mr. Allington will supervise sales of products manufactured by CF&I and its subsidiaries, including the recently acquired John A. Roebling's Sons Corp.

Donald Max Ashfal is now employed by Armco Steel Corp. in their research laboratories as a metallurgical assistant. He was previously with Caterpillar Tractor Co.

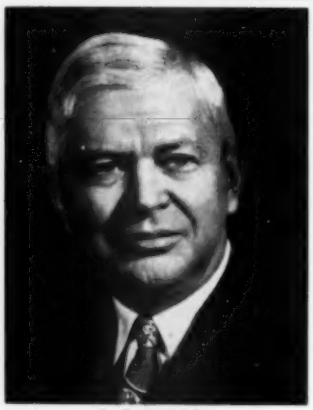
Robert A. Barnes is a mining engineer for the American Smelting & Refining Co. at their Silver Bell open-pit mine.

P. G. Bastien has been awarded the V. Noury Prize by the Institut de France (Academy of Sciences) for his contribution to metallographic research.

Clark B. Carpenter, head of the dept. of metallurgy at the Colorado School of Mines, has been appointed as dean of the Graduate School at the college. Professor Carpenter, who has served on the mines faculty since 1920, will continue his duties as head of the metallurgy dept. while also serving in his new position.

William G. Christy, until recently director, Bureau of Smoke Control, New York, has severed his connection with the city and opened an office as consulting engineer at 34 Park Row, New York.

Louis S. Deitz, Jr. is now technical advisor for Climax Molybdenum Co. of New York. He was previously with Nassau Smelting & Refining Co.



C. E. WILSON

Charles E. Wilson, an AIME member and former president of General Motors, is now Secretary of Defense in President Eisenhower's cabinet. Mr. Wilson graduated from Carnegie Institute of Technology as an electrical engineer. Shortly after his graduation he was employed by Westinghouse Electric & Mfg. Co. as a student apprentice. He remained in the employ of Westinghouse in the Pittsburgh district for following 10 years. During his early experience at Westinghouse he was student assistant to B. G. Lamme, chief engineer, and much of Mr. Wilson's later success and knowledge was credited to Mr. Lamme. In 1912 Mr. Wilson designed the first automobile starting motors made at Westinghouse. He joined General Motors in April 1919 as chief engineer and sales manager of the automobile division of the Remy Electric Co., a General Motors subsidiary. In 1921 he was made factory manager, in 1925 he was appointed general manager of the Remy Electric Co., and became vice president of General Motors in 1928. In 1939 he was made executive vice president of General Motors. From 1941 until he resigned to become Secretary of Defense he was president of the company.

W. S. Phillips has been promoted to the position of mine superintendent of the Holden mine, Howe Sound Co., Chelan Div.

Francis B. Speaker, deputy director, has succeeded **H. A. Sontag,** returned to Joy Mfg. Co., as director of the mining requirements div. of Defense Materials Procurement Agency.

ROBERT C. STEPHENSON

Robert C. Stephenson recently resigned as assistant state geologist of Pennsylvania to accept a position in the mineral division of Woodward & Dickerson, Inc., Philadelphia. He is currently eastern vice chairman of the Ind. Min. Div., AIME.

J. F. Frost, milling engineer for the Buchans Mining Co., Newfoundland, Canada, has been named mining engineer for western dept. of American Smelting & Refining Co. He succeeds **J. G. Piercy** who has been assigned to similar post with AS&R in N. Y.

Thomas K. Graham, superintendent of the Great Falls Zinc Plant, has been advanced to assistant general superintendent of the Great Falls Reduction Dept., Anaconda Copper Mining Co.

William F. Koch, district sales manager for Hercules Powder Co., since 1931, retired on Dec. 31 after 41 years in the explosives industry. He was succeeded by **Lawrence W. (Ben) Early,** formerly assistant manager of the Salt Lake district territory.

Walter S. Larsch, retired general manager of the Nevada Mines Div., Kennecott Copper Corp., has succeeded the late **Henry M. Rives** as a member of the Nevada State Tax Commission.

Frank N. Marr, president of the Spokane-Idaho Mining Co., has succeeded **E. C. Stephens** as president of the Northwest Mining Assn.

John S. Marsh of Bethlehem Steel Co. has been promoted from engineer to assistant chief of research. **J. J. Link** was promoted from assistant metallurgical supervisor, tin mills div., Sparrows Point plant, to assistant chief metallurgist at the plant.

D. A. RHOADES

D. A. Rhoades, Kaiser Aluminum & Chemical Corp., Oakland, Calif., was elected president of the Aluminum Assn. for the coming year. **A. V. Davis,** Aluminum Co. of America, New York was reelected chairman of the board and **Donald M. White** was reappointed secretary and treasurer.

M. G. Smerchanski, consulting mining geologist of Winnipeg, Manitoba, has been appointed as a member of the board of governors at the University of Manitoba. Mr. Smerchanski is the consulting geologist for the Lake Lingman Gold Mining Co., Ltd. as well as other mining companies.

J. A. Swart and Mrs. Swart passed through New York in February on their way to the Belgian Congo where he is to be manager for a transportation project of the Raymond Concrete Pile Co. Mr. Swart resigned as chief engineer for Foley Hermanos Ltd., at Chuquicamata, to accept the new post.

Mike Vincent is now employed by the Atlas Powder Co. as a sales engineer for the state of Nevada.

Boris S. Voukovitch has accepted a position of a geologist with the Geophoto Services in Denver, a geological consulting company.

John L. G. Weysser, formerly manager of the Lehigh Materials Co. at Lansford, Pa., and before that with the Lehigh Navigation Coal Co., has set up an office as consulting mining engineer. His new address is 712 Mahantongo St., Pottsville, Pa.

Hubert O. Woods is now working for the Allis-Chalmers Mfg. Co. at their main plant in Milwaukee as an erection engineer.

Harry J. Wolf left for Europe where he was to examine mining properties in Greece (Crete). On his return from Athens he planned to investigate mineral deposits in French Morocco and in Portugal.

Scott Turner, past president of the AIME, has just been elected president of the American Institute of Consulting Engineers.

F. J. TOKAR

F. J. Tokar is service engineer with the Quigley Co., Chicago.

Henry Mulryan has become president of the Sierra Talc & Clay Co., and the company has moved its offices from Los Angeles to South Pasadena.

Robert Sanford, mining engineer, whose last assignment was in Afghanistan, India, and Nepal, has been appointed special assistant to **Dr. C. W. Seibel,** regional director, Region VI, Bureau of Mines. He will supervise the Arkansas-White and Red river basin program and will serve on the Southwest Field Committee. He fills the position in Amarillo left vacant by the promotion of **Leon W. Dupuy** to the director's staff in Washington.

J. M. Ehrhorn was named head of new dept. of United States Smelting, Refining and Mining Co. to investigate and assist in the development of more efficient operating methods and the adaption of new methods in both mining and ores processing. **Boris Ashurkoff,** formerly mechanical engineer at one of the company's former Mexican subsidiaries, and since 1946 mechanical superintendent at the U. S. gold dredging operations at Fairbanks and Nome, Alaska, will be production development chief of the new project.

Herbert E. Fenton has been named Pittsburgh area mining sales and service engineer for the Carboloy Dept. of General Electric Co. Mr. Fenton was formerly associated with the Cardox Corp., Pittsburgh.

Fenton H. Finn, formerly vice-president, has been elected president of New York State Natural Gas Corp., a Pittsburgh company which wholesales gas to eight distributors including Peoples Natural Gas Co.

C. B. Foraker, acting mine superintendent, has been appointed mine superintendent of the Magma Copper Co., at Superior, Ariz.

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Obituaries

L. G. Johnson (Member 1917) died Nov. 16, 1952. He was born in Colorado and educated at the Colorado School of Mines where he received an E.M. degree. He held positions with Federal Lead Co. in Flat River, Mo. as superintendent of mines and was general superintendent of Federal Mining & Smelting Co. His major technical interests were mining, geology, and geophysics.

Vladimir Vasilievich (Member 1952) died Nov. 11, 1952. He was born in Belgrade, Yugoslavia. Mr. Vasilievich had been employed by Chicago Bridge & Iron Works for whom he worked in various parts of the United States and in Venezuela in the building of bridges. From 1928 up to the time of his death he worked for Cerro de Pasco Corp. in Peru. He was foreman on construction of concentrators, mine shafts, and power plants for this company.

T. K. Stinson (Member 1939) died Oct. 14, 1952. He had a long and varied career in the chemical field. Mr. Stinson worked for American Steel Foundries doing steel analyses, and was in charge of chlorine plant control laboratory for Monsanto Chemical Works. Mr. Stinson was also associated with Mathieson Alkali Works in Niagara Falls, N. Y., where he did analytical research in Synthetic Ammonia Dept. He was chief chemist in charge of Technical Dept. of Geo. S. Mepham Co. Born in Iowa, he was educated at Maryville College and Missouri University.

Frank Leslie Estep (Member 1921) died Dec. 11, 1952. One of his first jobs was chief draftsman for Cambria Steel Co. Upon leaving this company, he became associated with Tennessee Coal, Iron & Railway Co., first as assistant chief engineer and then as chief engineer. Mr. Estep was at one time vice-president of Perin Engineering Co. of New York City. From 1937 to 1941 he was engineer and then chief engineer for H. A. Brassert & Co. From 1943 up to the time of his death he was a consulting engineer and worked for various companies. During World War II he worked on defense plant projects and also did reports on Government-owned plants for the U. S. Assets Administration. Besides holding membership in AIME, he was a member of Assn. of Iron & Steel Engrs, Iron & Steel Institute, and American Society of Metals as well as many others.

Bert T. Macy, manager of the parts and accessories dept. of Sauerman Bros., Inc., Chicago, Ill., died at his home in that city on December 27 after a short illness. He was 66 years old. Mr. Macy had been with Sauerman Bros., Inc. for 31 years and had been a department head for most of that period. Previously he had been with the Chicago office of the American Steel & Wire Co.

Job H. Winwood (Member 1939) died Nov. 4, 1952. He was born at Coseley, England and attended the Michigan College of Mines where he received an E.M. degree. He was mining engineer with Silver King Coalition Mines Co. Mr. Winwood had a successful practice in engineering, mine examinations and reports and mining lease operations. His work had been confined to the U. S. and principally in Utah, Idaho, Arizona, Colorado, Oregon, Montana and Wyoming.

John L. Clarkson (Member 1942) died June 11, 1952. Mr. Clarkson was a mining engineer for the Superior Coal Co. for five years. From 1917 to 1939 he was connected with the Clarkson Coal Mining Co. and was general manager and president of this firm.

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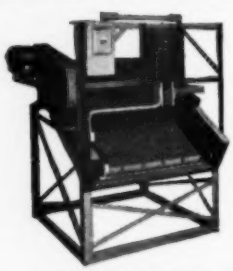

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G. TEMPLE BRIDGMAN

Appreciation of

G. Temple Bridgman

By H. DeWitt Smith

Grenville Temple Bridgman passed away at his San Francisco home on November 25, 1952. He was born in Newport, R. I. on August 7, 1881, went to Yale for a year and graduated from the Massachusetts Institute of Technology in 1909 with a B.S. degree. After the routine of exploration work in Mexico, Peru and the Philippines and a three-year tenure as manager of the Mascot Mine in Tennessee, he became assistant to Fred Hellmann, consulting mining engineer for Guggenheim Brothers in 1916.

This work took him to Braden and Chuquicamata and late in 1916 he served as acting assistant general manager at Chile Copper under Herman C. Bellinger. After returning to Mr. Hellmann's staff, he examined the tin mining districts of Bolivia before returning to New York in 1924 to become consulting mining engineer for Guggenheim Brothers, a position he held until 1940. During this period he directed the activities of the Pacific Tin Co., the only American dredging operation in Malaya, and became known as the American authority on tin and tin mining. In 1929-30 he served as president of the Mining & Metallurgical Society of America.

After retiring to San Francisco in 1940, he was called back to Washington in 1941 by the Reconstruction Finance Corp. as consultant on erection of an American tin smelter, and soon after accepted the position of executive vice-president, Metals Reserve Co., in charge of Government purchases of critical and strategic metals and minerals. In this key position, in spite of failing health, he gave unsparingly of his strength seven days a week from 1941 to 1944. He then assisted Will L. Clayton as deputy Surplus Property Administrator before returning to San Francisco.

In organizing Metals Reserve Co. for war service, Temple Bridgman gathered around him a group of outstanding mining engineers and lawyers, and put them on their own responsibility in making contracts, subject to his wise and far-sighted guidance. He did this so successfully that in 1943, Harold D. Smith, then director of the budget, advised the R.F.C. that he could not see how its subsidiary, Metals Reserve Co., could handle properly the volume of business it was doing with so small a staff.

Greatly impressed by the long drawn-out litigation connected with Government purchases of metals and minerals in World War I, he saw to it that every Metals Reserve procurement contract issued included a fair cancellation clause. The Contract Settlement Board, after its experience in terminating Army and Navy contracts, would not believe until shown, that Metals Reserve contracts offered no problems requiring its services. So fairly and wisely were these contract terminations drawn under Temple Bridgman's direction that no successful suit has since been brought against Metals Reserve.

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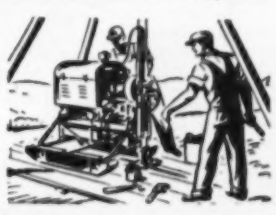

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He reserved to himself the difficult task of negotiations with the Board of Economic Warfare and the Department of State, some of whose staff seemed bent on making two dollars do the work of one, particularly in Latin America. Time after time, when tempers flared to the explosion point, Temple flashed through with a pertinent story or bit of dry humor to relieve tension and permit the reaching of a sound decision.

Pope Yeatman, dean of American mining engineers, in recalling our friend's work in South America, writes "Temple Bridgman has left with us the memory of a fine engineer, who in his long career demonstrated the integrity, loyalty and vision which we like to believe is the hallmark of our profession."

Temple Bridgman will long be remembered by his devoted family, friends, and associates, for his kindness, humor, wisdom and integrity. His soul goes marching on.

Appreciation of

H. M. Lavender

by the Board of Directors

of the Arizona Section, AIME

Harrison M. Lavender (Member 1939), vice-president of Phelps Dodge Corp., died on Mar. 21, 1952. Born in Scotland, S. D. in 1890, Mr. Lavender spent his boyhood in Colorado and graduated from the Colorado School of Mines in 1916 with the degree of mining engineer. He then became engaged in metal mining work in the Western States until November 1917 when he enlisted in the U. S. Army and served as an enlisted man and officer in the Engineer Corps for the duration of World War I.

Mr. Lavender entered the employ of the Calumet & Arizona Mining Co. in Bisbee, Ariz. in 1924 as a miner and successively held the positions of shift boss, mine foreman and chief engineer for that company. At the time of the merger with Phelps Dodge Corp. in 1931 he became mine superintendent of the Copper Queen Branch of Phelps Dodge at Bisbee. Mr. Lavender continued in that capacity until 1935 when he was transferred to the Western General Offices of Phelps Dodge at Douglas, Ariz. as assistant to the vice-president and general manager. In 1937 he became general manager of the Phelps Dodge Metal Branches and in 1946 became vice-president and general manager. He was made a director of Phelps Dodge Corp. in 1949. Due to ill health he relinquished the title of general manager on Mar. 1, 1952 but continued as a vice-president to the time of his death.

During Mr. Lavender's period of service Phelps Dodge Corp. undertook a large expansion program in which he had a prominent part. It was under his immediate direction that the large Morenci development was brought into successful production and became Arizona's greatest copper mine. He then undertook the modernization of the Company's plant at Ajo, Ariz., including the modern electric haulage system and construction of a new smelter. He was responsible for planning the development of the new open-pit project at Bisbee which is now called the Lavender Pit in his honor. Mr. Lavender was a great mining engineer and won distinction as an outstanding executive. In his passing Arizona and the entire mining industry suffered a great loss.

He was awarded the Medal for Distinguished Achievement by his alma mater, Colorado School of Mines, in 1948. He was a director of the AIME and served as chairman of the Western Div. of the American Mining Congress in 1947.

Harry Lavender, the name by which he was generally known, was an outstanding athlete and star football player at the Colorado School of Mines. He was an earnest supporter of baseball in the Southwest, and his counsel was sought by many of the outstanding names in the sport world.

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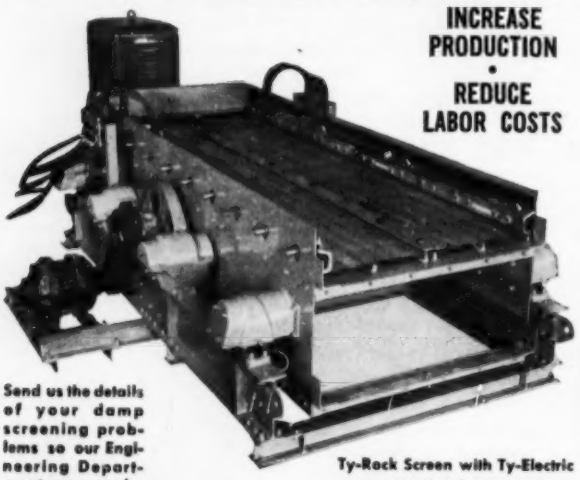

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NECROLOGY

Date Elected	Name	Date of Death
1921	A. W. Ambrose	Dec. 28, 1952
1925	Charles H. Herty, Jr.	Jan. 17, 1953
1938	George W. Levangie	Jan. 7, 1953
1915	W. L. Maxson	Jan. 8, 1953
1943	Halbert M. McClain	Unknown
1936	Maxwell N. Short	Unknown
1929	Peter Trotzig	Unknown

Proposed for Membership

MINING BRANCH, AIME

Total AIME membership on Dec. 31, 1952 was 18,643; in addition 1,964 Student Associates were enrolled.

ADMISSIONS COMMITTEE

T. D. Jones, Chairman; Thomas G. Moore, Vice-Chairman; H. S. Bell, F. W. Hanson, R. H. Chadwick, T. W. Nelson, C. A. R. Lambly, John T. Sherman. A. C. Brinker, G. P. Lutjen, Ivan Given, E. A. Prentis, C. Leslie Rice, Jr., and J. H. Scaff.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

Alabama
Birmingham—Younger, Joseph J. (C/S—S-J)
Hueytown—Hutchings, Thomas E. (M)

Arizona
Cottonwood—Stotts, John L. (C/S—S-J)
Douglas—Heuring, Francis T. (C/S—S-J)

California
Bishop—Lyons, Mark S. (C/S—S-J)
Bishop—Wright, Lawson A. (R. M)
Burlingame—Kramm, Douglas J. (C/S—A-M)
Chico—Pflug, Gordon E. (C/S—S-J)
Colton—McCormick, Earl L. (C/S—S-A)
Long Beach—Buster, John L. (R. C/S—S-J)
Monrovia—Kennedy, John J. (M)
Nevada City—Siegfried, Joshua F. (M)
Pasadena—Johnsen, Eugene R. (M)
Riverside—Walls, Marion B. (M)

Colorado
Denver—Kautz, Harry P. (C/S—J-M)
Glenwood Springs—Hart, Howard R. (A)
Grand Junction—Kay, Marvin L. (M)
Grand Junction—Morgan, James E. (A)
Grand Junction—Outcalt, Earl E. (C/S—S-J)

Connecticut
Westport—Marston, William T. (C/S—J-M)

Florida
Lakeland—Clarke, Charles M. (M)
Lakeland—Olsen, Clarence G. (R. C/S—S-M)

Idaho
Kellogg—Cote, Ralph W., Jr. (C/S—S-J)
Kellogg—Mize, Robert J. (R. C/S—J-M)
Stibnite—Mahoney, Benjamin F. (M)
Wallace—Crosby, Garth M. (C/S—A-M)

Illinois
Dowell—Williamson, Harry, Jr. (J)
LaSalle—Miller, Pierce J. (C/S—A-M)
Northbrook—Weir, John P. (C/S—A-M)
Peoria—Grenaway, Nicholas J., Jr. (C/S—S-J)

Indiana
Corydon—Pullen, Leslie W. (C/S—A-M)

Kansas
Norton—Mallette, Charles M. (C/S—S-J)

Kentucky
Wheelwright—Jimenez, Ricardo A. (R. C/S—S-J)

Maryland
Arnold—Ames, John A. (C/S—J-A)
Hyattsville—Bertoni, Albert P. (J)
University Park—Panek, Louis A. (C/S—S-J)

Michigan
Calumet—Campbell, Colin A. (M)
Lake Linden—Miller, Emory W. (C/S—S-J)

Minnesota
Calumet—Livingston, Richard W. (J)
Eveleth—Hartley, David (A)
Hibbing—Blake, Rolland L. (R. C/S—S-J)
Minneapolis—Conrad, Malcolm A. (C/S—S-J)
Virginia—Blackwell, Douglas C. (C/S—S-J)

Missouri
Rolla—James, Jack A. (C/S—A-M)
St. Louis—Cox, Willard E. (C/S—S-J)
Webb City—Kirk, Whitson J. (C/S—A-M)

Montana
Butte—Fall, Samuel C. (C/S—S-J)
Butte—Hammer, Donald F. (C/S—S-J)
Butte—Hill, Robert B. O. (C/S—S-J)
Butte—Johnson, Henry F. (M)
Butte—Madelung, Robert R. (C/S—S-J)
Butte—Shepherd, R. Ernest (C/S—S-J)
Great Falls—Balhiser, Robert H. (M)

Nebraska
Scottsbluff—Layman, Frederic G. (C/S—S-J)

Nevada
Boulder City—DeMoss, Edwin C. (C/S—A-M)
Gabbs—Reinmiller, Frank W. (R. M)

New Mexico
Carlsbad—Edmunds, James (R. C/S—J-M)
Carlsbad—McGuire, Benjamin F. (C/S—A-M)
Hanover—Tipton, William D. (C/S—S-J)
Roswell—Moore, Charles E., Jr. (C/S—S-J)

New Jersey
Metuchen—Sigurdson, Harold (M)
Orange—Cole, Allan A., Jr. (C/S—S-J)
Phillipsburg—Wickens, Charles D. (C/S—S-J)

New York
Balmat—Cropper, Wallace J. (J)
Flushing—Nininger, Robert D. (C/S—A-M)
Gouverneur—Miller, Roger N. (C/S—S-J)
Manhasset—McKay, Cameron E. (R. M)
Port Washington—Dudley, James C. (C/S—J-M)

North Carolina
Raleigh—Nixon, Edward C. (J)

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Pennsylvania
Beaver—Mitchell, Robert F. (R. C/S—J-M)
Bethlehem—Anderson, Frank J. (J)
Bridgeville—Horak, George C. (R. C/S—S-M)
Catasauqua—Douglass, Alfred E., Jr. (M)
Clearfield—Moore, Philip C. (C/S—S-J)
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Coming Events

Mar. 8-11, American Institute of Chemical Engineers, Buena Vista Hotel, Biloxi, Miss.

Mar. 11, AIME, Connecticut Section, Brass Mill Clinic, Waterbury, Conn.

Mar. 16-20, National Assn. of Corrosion Engineers, annual conference and exhibition, Hotel Sherman, Chicago.

Mar. 23-27, ASM, Western Metal Congress and Exposition, Pan-Pacific Auditorium, Los Angeles.

Apr. 12-16, Electrochemical Society "Symposium on the Application of Electrochemistry to Biology and Medicine," Hotel Statler, N.Y.C.

Apr. 12-May 23, Empire Mining and Metallurgical Congress, Australia-New Zealand.

Apr. 20-22, AIME, National Open Hearth and Blast Furnace, Coke Oven and Raw Materials Conference, Hotel Statler, Buffalo.

April 20-22, 1953 Metal Powder Show and Ninth Annual Meeting, Hotel Cleveland, Cleveland.

Apr. 25-May 10, Fifth Liege International Fair, Liege, Belgium.

May 11-14, American Mining Congress Coal Convention & Exposition, Public Auditorium, Cleveland.

May 14-16, Pacific-Northwest Metals and Minerals Conference of 1953, joint meeting of Metals Branch and Industrial Minerals Div., Ben Franklin Hotel, Seattle.

May 14-23, Thirtieth Anniversary International Petroleum Exposition, Tulsa.

May 18-22, Fifth National Materials Handling Exposition, Convention Hall, Philadelphia.

June 3-7, Chemical Institute of Canada, Windsor, Ont.

June 16-19, Materials Conference held concurrently with First Exposition of Basic Materials, Hotel Roosevelt, New York.

Sept. 8-12, Joint Meeting Industrial Minerals Division, AIME, Keltic Lodge, Ingonish, Nova Scotia.

Sept. 21-23, American Mining Congress Mineral Mining Convention, Olympic Hotel, Seattle.

Oct. 19-21, Institute of Metals Division, Fall Meeting, Hotel Allerton, Cleveland.

Oct. 28-31, AIME, El Paso Meeting, in cooperation with International Mining Days, Paso Del Norte, El Paso.

Oct. 29-30, AIME, ASME Fuels Conference, Conrad Hilton Hotel, Chicago.

Oct. 29-31, Annual Meeting of National Council of State Board of Engineering Examiners, Plaza Hotel, San Antonio.

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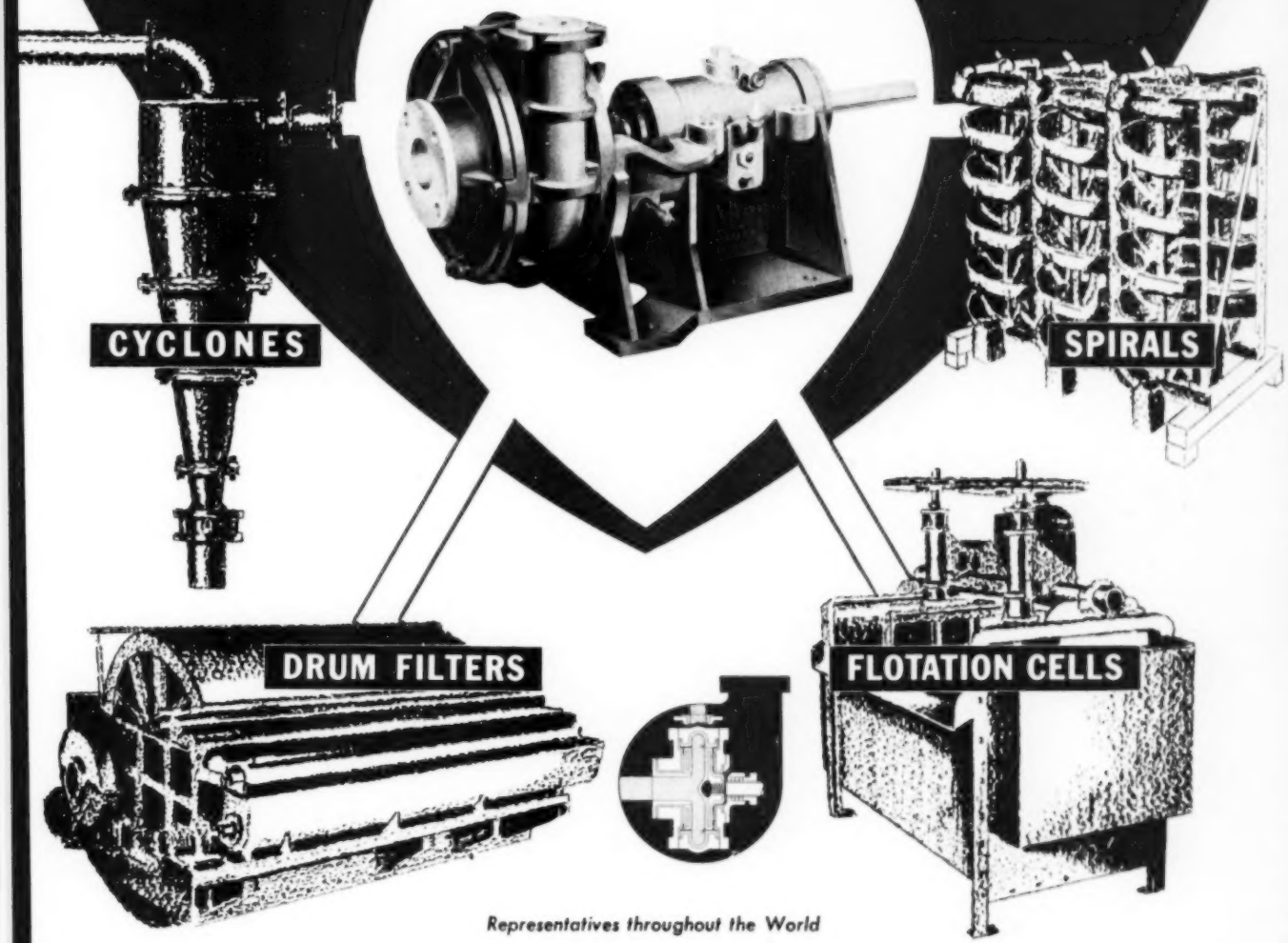

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